COAL MINING

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AN ELEMENTARY TEXT-BOOK OF COAL MINING

A CLASS-BOOK FOR ELEMENTARY STUDENTS PREPARING FOR THE BOARD OF EDUCATION EXAMINATION IN "PRINCIPLES OF MINING" AND FOR COLLIERY MANAGERS' EXAMINATIONS

BY

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PREFACE

This book was first published in September, 1893. The task of revising a book written by another author is at all times difficult, and in the case of a work that has already gone through nineteen editions it is rendered more so by the fact that the author has already made so successful an appeal to those interested in his subject.

Coal Mining, however, is progressive, and new methods and appliances are constantly replacing old, so that no matter how successful a book on the subject may have been, it can only remain so by being brought carefully up to date.

In the present edition this has been done, and a considerable amount of new matter has of necessity been incorporated. The general arrangement has been adhered to, and, as far as possible, the views of the author have been retained. Many arithmetical examples have been inserted, and it is hoped that they may add to the usefulness of the book.

It is the reviser's hope that the established utility of the book has been maintained, and that Peel's *Coal Mining* may continue to retain its already well-established place in the literature of Mining.

D. B.

June, 1921.

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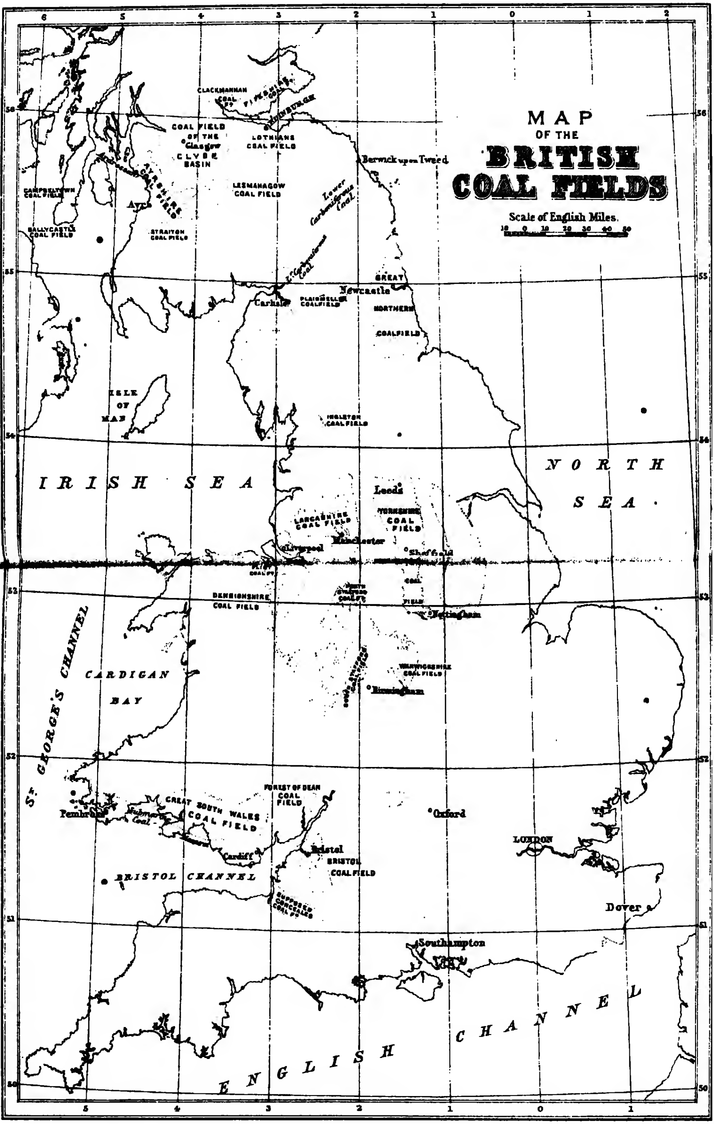
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COAL MINING

CHAPTER I

MINERAL GEOLOGY

Value of a Knowledge of Geology to Mining Students.—Although the subject of this book is coal mining, yet it is necessary in the first place to enter a little into the subject of Geology so far as it relates to Coal Mining, because an elementary knowledge of this science is of great importance to mining students, if they wish to properly understand the various operations in mining.

A knowledge of *geology*, with its sister science *mineralogy*, will help us to recognize the various valuable minerals to be found in the crust of the earth, to decide where we may search for them with a reasonable prospect of success, and where it would be utterly futile to explore. It will also explain to us the origin of coal and the processes by which it has been formed, and how the faults, dykes, and other interruptions that are encountered in the workings of mines have occurred.

To the explorer for minerals, a knowledge of the facts and principles of geology is requisite, in order to ascertain the character of the rocks over which he passes and to assign to them their proper position in the geological scale. If he is searching for the Coal Measures of the Carboniferous System, he should be able to determine

their relative position from an examination of the rocks which appear at the surface, and to say whether there is any probability of coal being found at a workable depth below the surface. Many instances have occurred in this country of large sums of money being expended in prosecuting exploring operations, such as boring and sinking, in rocks below the coal measures, which a slight knowledge of geology, or that branch of geology known as *palæontology*, would have prevented.

Unstratified and Stratified Rocks.—The rocks composing the " crust of the earth ", or that portion of

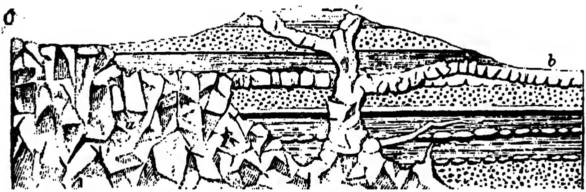


Fig. 1.—Igneous Rocks, *a*, *c*, *b*, breaking through and intruding on Stratified Rocks. The igneous rock at *c* breaks through the sandstones, shales, &c., nearly at a right angle; but the intrusive band *b* has forced its way laterally, and lies almost parallel to them.

the solid structure of the earth that we are able to examine and investigate, namely about 20 miles in vertical thickness, are divided by geologists into *unstratified* and *stratified* rocks.

Unstratified rocks have been originally thrown up in a molten condition from the interior of the earth, and have cooled down and become hardened. They are frequently termed "*igneous*" rocks. They occur in *amorphous* or shapeless masses, and frequently fill cracks and fissures in the stratified rocks into which they have been forced from below (see fig. 1). They also occur interstratified with stratified rocks. Much change has often been produced in the stratified rocks owing to the highly heated state of the material composing the unstratified rocks when it was forced upwards from the

interior of the earth. Thus, coal for a considerable distance on each side of a basaltic or whin dyke (a rock of this class) is usually altered to a hard dense mass resembling coke. Adjoining shales and sandstones are also altered by the effects of the heat.

Unstratified or igneous rocks may be known by their extreme density and hardness, by their amorphous shape, by the frequent manner of their occurrence as masses intruded through beds of stratified rock, by their usually crystalline texture, and by the entire absence in them of plant and animal remains. Granite, basalt, syenite, and diorite are examples of unstratified rocks.

Stratified rocks are those which occur in parallel layers, beds, or strata. They are also known as *sedimentary* or *aqueous* rocks, because they have for the most part been formed under water by deposition as sediment. The matter now constituting such rock masses is believed to have been borne, in the form of sand and mud, by streams to seas or lakes and there deposited.

The process by which stratified rocks have been formed may be seen going on wherever there is running water. Every stream bears down in its current a certain amount of gravel, sand, and mud. When the velocity of the current is checked and stopped on reaching the sea or lake into which the river flows, the matter thus borne along by the water sinks to the bottom. The heavier particles fall first, the lighter particles next, and finally the very fine particles of mud, and thus are formed beds of gravel, sand, and mud. Such beds, deposited by streams flowing in long bygone ages, become solidified in the course of time through pressure of material deposited above them, the action of the internal heat of the earth, and the deposition between the grains of binding materials such as iron, lime, and silica. These now form rock masses; and just as we find in river deposits the regular alternation of gravel, coarser sand, sand, and mud, so we find in the rocks the alternation of conglomerate and coarse-grained sandstone, then sandstone and shale.

The animal remains known as *fossils*, which are found so numerously in stratified rocks, are the petrified remains of creatures that inhabited the waters in which the rocks were formed, or of remains which were by chance deposited there. They give very strong proof of the aqueous origin of such rocks. But there is still further proof. We find imprinted on the rocks *rain-marks*, *footprints*, *sun-cracks*, and *ripple-marks*, which show that the now solid rocks were once in a soft, plastic condition, such as mud and sand.

Certain stratified rocks have, however, been formed in a somewhat different manner from this. For instance, chalk and many limestones are organic in their origin, being chiefly composed of the shelly remains of creatures which lived in ancient seas; and a certain kind of limestone has been formed chemically, having been produced by the precipitation of carbonate of lime in an inland sea or lake, the water of which had become *supersaturated* or overcharged with this substance.

Coal also, though occurring as a stratified rock, is of organic origin, having for the most part been formed by the decay of vegetable matter on land, or low-lying swamps and shallow estuaries.

Relative Age of Rocks.—In stratified rocks occurring in layers or beds, the bed lowest in a series may be considered of greater age than the uppermost bed. We cannot precisely determine, in years, the antiquity of any bed or series of beds, but their relative ages have been in a measure ascertained, and form a basis of arrangement or classification. Stratified rocks (about twenty miles in vertical thickness) have thus been divided by geologists into about a dozen formations or systems, according to their *lithological* and *palæontological* character, which it has been possible to arrange according to relative age.

Formations are always found to occupy the same relative position in the crust of the earth—thus, although a given formation may occur at a great depth in one

part of the country and at the surface in another, yet where it occurs in conjunction with other formations it is always found to occupy the same relative position in regard to them. A formation that occurs below another is not found above it in any other place, except it be as a result of some great disturbance of the strata, of which ample evidence is in such cases generally present. Thus we may say that in the whole series of sedimentary rocks there exists this order, which is termed the "order of superposition".

The fossils found in the rocks have enabled geologists to group formations according to the characteristics of the fossils they contain. Each formation has fossils peculiar to itself, the more remote the formation the more unlike existing animals and plants are the fossils found to be, and the lower their position in the scale of life.

Division of the Stratified Rocks into Groups and Formations.—Stratified rocks are arranged, according to the general characteristics of the fossil remains contained in them, into three great groups:

3. Cainozoic or Tertiary (recent life).
2. Mesozoic or Secondary (middle life).
1. Palæozoic or Primary (ancient life).

Each of these groups is subdivided into the formations or systems noted below, and it is most remarkable that in so small an area as that occupied by the British Isles, every system is more or less represented.

CAINOZOIC	Post-Tertiary or Quaternary.	
	Tertiary proper.	
MESOZOIC	Cretaceous.	
	Jurassic	Oolite.
		Lias.
	Trias.	
PALÆOZOIC	Permian.	
	Carboniferous.	
	Devonian and Old Red Sandstone.	
	Silurian.	
	Cambrian.	
	Archæan.	

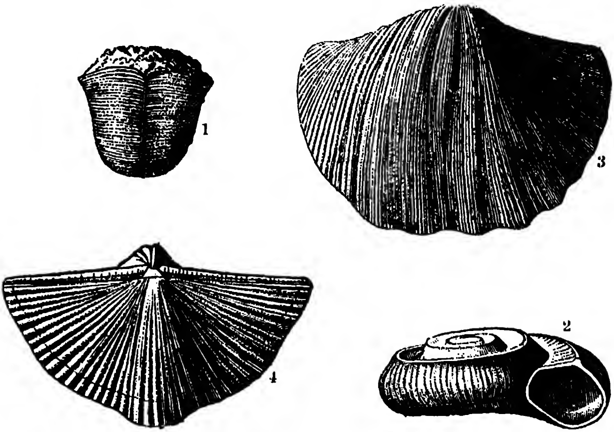
The Carboniferous Formation.—Each of these formations admits of further subdivision, but as we are entering upon the subject of geology only in so far as it directly bears upon coal mining, we will confine our attention to a brief consideration of the *Carboniferous System*, which contains the true coal measures. It is subdivided as follows:

CARBONIFEROUS FORMATION	Coal Measures.
	Millstone Grit.
	Carboniferous Limestone.

This formation is called *Carboniferous* because of the remarkably large amount of carbon it contains. It consists largely of three kinds of rock, namely *coal*, *shale*, and *carboniferous limestone*. The coal contains about four-fifths of its weight of carbon, the shales often contain a certain amount, and the limestone also a little. The formation is, therefore, appropriately named "carboniferous".

Carboniferous Limestone.—After the deposition of the Old Red Sandstone, upon which the carboniferous rests comformably, the sea bottom became depressed, and, in the deeper waters, the Carboniferous Limestone was deposited. This limestone now forms many high hills and deep valleys in the counties of Somersetshire, Derbyshire, Yorkshire, and Durham, through which it runs, and it is frequently termed the *Mountain Limestone*. It is about 3000 feet thick, mainly composed of corals, shells, and encrinites, and is of a light-grey colour. Four of the most characteristic shells it contains are shown in figs. 2–5. In north-west Yorkshire thin beds of sandstone and shale are found intercalated with the limestone, and are known as the "Yoredale Rocks". Travelling northwards, these beds get thicker and more numerous, and the limestone thinner. In the north of England and in Scotland the shales become carbonaceous, and occasional beds or seams of coal appear.

The *Millstone Grit* is variously known to miners by the terms " rough rock ", " moor rock ", and " farewell rock ". The latter is applied because no coal is found beneath it, except in the north of England and the east of Scotland, where all the principal seams of the Fife and Lothians coalfields lie in the Calciferous Sandstone Series, which is under the Millstone Grit. The Mill-



Figs. 2–5.—(1) *Bellerophon hiulcus*, and (2) *Euomphalus pentangulatus* two gasteropod shells; (3) *Productus giganteus*, and (4) *Spirifer striatus*, two brachiopods.

stone Grit is a very coarse sandstone, and is found in thick and massive beds. It is sometimes quarried for millstones, whence its name, and for building purposes.

The *Coal Measures*, in which we are chiefly interested, are found resting upon the Millstone Grit, and in them are found the coal beds or seams. The total thickness of the coal measures varies from 3000 feet in Northumberland to about 12,000 feet in South Wales, and they consist of alternations of beds of sandstone, shale, ironstone, and coal-seams. The latter occur at various distances apart, and vary from less than an inch to several

feet in thickness; the thickest seam of coal in the British Isles is the famous " Ten-yard Seam " of South Staffordshire.

Origin of Coal.—After the deposition of the Millstone Grit, the sea bottom appears to have been frequently elevated, and become dry land, upon which a rich and luxuriant vegetation flourished, favoured by a warm and moist climate. This vegetation spread rapidly, and eventually the land was covered with thick, dense forests. Then, submergence taking place, the vegetation was covered by water; and beds of mud and sand, now shale and sandstone, were deposited upon it. Elevation and depression recurred at frequent intervals, and now each seam of coal in the coal measures represents an elevation and depression of the land. In this way the coal measures, consisting of beds of shale, sandstone, and coal-seams, have been formed.

Coal is therefore mineralized vegetable matter, or the result of the decomposition of vegetable matter. As a proof of the vegetable origin of coal, distinct traces of woody tissue or the spores of ferns can sometimes be detected in thin slices of coal. The under-clay, found generally beneath seams of coal, is believed to have been the soil upon which the vegetation now converted into coal flourished. It is better known to miners as " seggar-clay " and " fire-clay ". It is usually full of fossilized roots of various plants, and in some instances fossilized stumps of trees have been found, standing upright as when they grew, passing through the coal into the rock above, with the roots spreading among the underclay. Another proof is afforded us in the chemical composition of coal, which is what might be expected to result from vegetable matter having been subject to great pressure, heat, and chemical action. In wood the chemical composition is about 50 per cent of carbon, 6 per cent of hydrogen, and 44 per cent of oxygen. These percentages are altered, step by step, through the various gradations of ~~*wood, peat*~~, and *lignite*—until the

carbon reaches about 85 per cent, hydrogen 4 per cent, and oxygen 11 per cent in bituminous coal.

The roof of coal-seams, especially when of shale, is often covered with the impressions of ferns and other plants which have contributed to the coal-seam. (See fig. 6.)

Coal-basins.—The coal-fields of Great Britain occur in basin-shaped depressions. It is generally understood

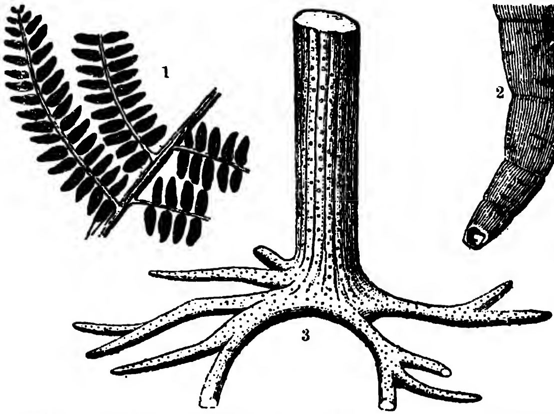


Fig. 6.—(1) *Neuropteris gigantea*, a fern; (2) *Calamites cannæformis*, a "horse-tail"; and (3) *Sigillaria* (a lycopod), with its roots (known as *Stigmaria*) attached.

that the Carboniferous Formation originally extended over nearly the whole of England, the south part of Scotland, and a considerable part of Ireland. But it has undergone many changes; the land has been upheaved and depressed, and folds or bends, termed *synclinals* and *anticlinals*, have been produced (fig. 9). Anticlinal ridges, running roughly parallel east and west, and others running north and south, as for example the Pennine chain, have been formed, and again been worn down, or denuded, till only the coal measures in the

hollows or basins, some of which have been covered by more recent formations, have been preserved as detached basins. Ireland has not been so fortunate in this respect as Britain, the coal measures having been almost entirely worn away by denudation.

The chief coal-fields of the British Isles are:

The Northern Coal-field.—This field lies partly in the county of Northumberland and partly in Durham, and extends from the River Coquet to the River Tees. It is about 50 miles long from north to south, with a width of from 5 to 30 miles.

The Cumberland Coal-field.—This field lies along the coast and extends from Whitehaven to Wigton; its length is about 25 miles and its width about 5 miles.

The Midland Coal-field.—This field lies between Nottingham and Leeds; its length is roughly 66 miles and its width from 10 to over 30 miles.

The Lancashire Coal-field.—This field has a very irregular outline and stretches from Knowsley to Ashton, a length of about 32 miles, the width being from 6 to 10 miles.

The North Wales Coal-field.—In this part the coal measures stretch from Oswestry to the mouth of the Dee, a distance of about 44 miles; their greatest width is about 4 miles, but in many places the field is narrower and much broken up.

The Leicestershire Coal-field.—This field forms a rectangular area of about 24 sq. miles with Ashby-de-la-Zouch as the centre, but there is a considerable stretch of concealed coal measures lying to the south-east and to the west.

The Warwickshire Coal-field.—This field extends from Tamworth to Warwick, a distance of 22 miles, and has a width of about 8 miles at its widest part near Bedworth.

The South Staffordshire Coal-field.—This field extends from Brereton to Bromsgrove-Lickey, a distance of about 26 miles, its width in the vicinity of Wolverhampton being about 9 miles.

The Shropshire Coal-field.—This is really a broken area containing the Shrewsbury, Coalbrookdale, and Forest of Wyre coal-fields; it extends with minor interruptions from Shrewsbury to the River Teme.

The Bristol and Somerset Coal-field.—This coal-field lies to the north-east of Bristol, the extreme length exposed being about 25 miles, but it disappears under the Secondary Rocks of the southern counties where it forms the recently discovered Kent Coal-field, and, passing under the Channel, reappears in the north of France and Belgium.

South Wales Coal-field.— This, the finest of our coal-fields, extends from Newport to St. Bride's Bay, a distance of about 90 miles. Its maximum width is in Glamorganshire, where it reaches 20 miles, but its average width is about 15 miles until it stretches beyond Swansea Bay, when it narrows down to a strip of about 3 miles wide.

Forest of Dean Coal-field.—This field is generally looked upon as an outlier of the South Wales field. It forms a perfect basin and is situated around the towns of Coleford and Cinderford in Gloucestershire. Its area is about 34 sq. miles.

The Scottish Coal-fields.—What is known as the Central Valley of Scotland consists of a large tract of low-lying ground stretching right across the country. Its northern margin is formed by the Highland Boundary Fault, running in a line between Greenock and Stonehaven, and its southern one by the Upland Boundary Fault, running from Girvan to Dunbar. This area contains the Lanarkshire, Ayrshire, Fifeshire, and Haddington Coal-field, with an aggregate coal-bearing area of about 800 sq. miles. On the western side the Lanarkshire and Ayrshire fields are of Carboniferous Age, while on the east, the Fife and Haddington fields are of Carboniferous Limestone Age.

Ireland is very deficient in coal resources. The most extensive fields are found in the counties of Kilkenny,

Queen's County, and Carlow. The greater part of the coal is culm or anthracite.

The output of coal from these fields, the number of underground workers, tons produced per man per annum from 1913 to 1919 inclusive, are given in the following table:

Year.	Workers Underground.	Total Tons Produced.	Tons per Man.	Deaths per 1,000,000 tons due to Accidents.
1913	909,834	287,411,869	316	5·81
1914	915,381	265,643,030	290	4·37
1915	754,673	253,179,446	335	4·9
1916	792,911	256,348,351	323	4·92
1917	811,510	248,473,119	306	5·27
1918	794,843	227,714,579	287	5·86
1919	954,806	229,743,128	240	4·67

For the ten years from 1903 to 1912 the death-rate due to accident was 4·76 per million tons of coal raised. It is therefore evident that in spite of shorter hours, the New Mines Act, and orders galore, mining is still as dangerous an occupation as ever. Legislation has failed to decrease the death-rate; we should now try education.

Concealed Coal-fields.—Great as are our proved resources of coal, they are not inexhaustible. In many districts the thickest, most valuable, and most cheaply worked coal-seams are, in fact, being rapidly exhausted. It is of national importance and interest that search be made for any extension of known coal-fields under newer formations, or for new coal-seams in hitherto partially explored districts.

The national importance of the question of coal supplies resulted in the appointment, in 1901, of a Royal Commission "to inquire into the extent and available resources of the coal-fields of the United Kingdom; the rate of exhaustion which may be anticipated, having

regard to possible economies in use, by the substitution of other fuel or the adoption of other kinds of power; the effect of our exports of coal on the home supply, and the time for which that supply, especially of the more valuable kinds of coal, will probably be available to British consumers, including the Royal Navy, at a cost which would not be detrimental to the general welfare; the possibility of a reduction in that cost, by cheaper transport or by the avoidance of unnecessary waste in working through the adoption of better methods and improved appliances, or through a change in the customary term and provisions of mineral leases; and whether the mining industry of this country, under existing conditions, is maintaining its competitive power with the coal-fields of other countries ".

The Royal Commissioners devoted a good deal of attention to the question of coal areas in the course of their investigations, and elicited much valuable information. The results of the inquiry were published in 1905, and showed that down to a depth of 4000 feet the available resources of the country in seams of 1 foot thick and upwards are as follows:

	Tons.
Estimate in proved coal-fields ..	100,914,668,167
Estimate below 4000 feet depth in proved coal-fields	5,239,433,980
Estimate of coal in concealed and unproved coal-fields within a depth of 4000 feet	39,483,000,000
Total	145,637,102,147

In recent years the known areas of the various coal-fields have been considerably increased by the discovery that the seams extend under newer formations beyond the limits previously judged.

One of the most important discoveries is that of the eastern extension of the Yorkshire and Nottingham coal-fields. Bore-holes and shafts have proved the existence of the Barnsley and other seams at a depth of 3000 feet

over a very large additional area. In Staffordshire also the area of the coal-field has been considerably enlarged by recent discoveries of the seams at great depths. In Northumberland and Durham coal is being won in larger quantities from under the sea. Many seams have been found in Kent, and coal is now being worked.

Industrial Products of the Carboniferous Formation.—Rich as the Carboniferous Formation is in coal, it also yields other valuable products.

Fire-clay.—This is generally found forming the floor of coal seams, and is variously termed "under-clay", "seggar-clay", and "fire-clay". It is assumed to have been the soil upon which the vegetation now converted into coal flourished. It is usually full of fossilized roots of various plants, but chiefly stigmaria of sigillaria.

Forming the floor of coal-seams, it is largely mined in conjunction with coal, especially thin seams, and is used for making fire-clay goods, such as bricks, gas retorts, &c., which will resist a very high temperature without melting or becoming soft.

It is generally of a grey colour, is composed chiefly of silica and alumina, and frequently contains nodules of ironstone. About three million tons of it are worked annually in the United Kingdom.

ANALYSIS OF FIRE-CLAY

Silica	51·10
Alumina	31·35
Oxide of Iron	4·63
Magnesia and Lime	3·00
Water and Organic Matter	10·47

Ganister.—This is a very hard, fine-grained kind of under-clay, usually full of stigmarian roots, and is found chiefly in the North of England. It is worked for making special bricks for lining blast-furnaces, coke-ovens, and other places where there is intense heat.

Ironstone.—Associated and interstratified with coal-seams are beds of ironstone. Some of these beds are of

sufficient thickness and good quality to be worth working, either in conjunction with the coal-seams or independently. When it is composed of carbonate of iron with a clay matrix it is called clayband or clay ironstone, and when there is a large proportion of carbonaceous matter it is called blackband ironstone.

A good deal of this kind of ironstone is worked from the coal measure shales.

The Carboniferous Limestone is largely quarried to burn into lime, and also to be used as a flux in iron smelting.

Veins of lead ore, zinc ore, and occasionally copper ore are found in it.

Other products, such as barytes (heavy spar), building stone, millstones, and flagstones are yielded by the Carboniferous Formation.

Coal in other Formations.—Although the Carboniferous Formation contains the richest and most abundant supply of coal, yet it must not be supposed that it is the only coal-producing formation, for, as a matter of fact, coal is found more or less in nearly every formation.

Thus, from the *Silurian*, anthracite has been worked in Ireland and Portugal; from the *Devonian*, bituminous coal has been worked in the north-west of France; the *Permian* yields bituminous coal in Bohemia and North America; the *Trias* yields a large quantity of bituminous coal and lignites in India, Germany, and Australia; the *Jurassic* yields the Kim coal of Kimmeridge, the moorland coal of Yorkshire, and lignites in Hungary and Austria; the *Cretaceous* contains lignites on the Continent and in New Zealand; the *Tertiary* yields lignites and peat; in our own country lignite has been worked for a long time at Bovey Tracey, as well as on the Continent.

Bituminous or Oil Shale.—Shale is consolidated mud or clay, altered by time and pressure. Mud, which at the time of its deposition was much mixed with car-

bonaceous matter, sometimes occurs in large quantities, and is called *carbonaceous*, *bituminous*, or *oil shale*. Many of these oil shales contain as much as 80 to 90 per cent of mineral matter, and seldom more than 20 per cent of volatile matter. Such shales are found in several formations, and they are largely worked in some parts, and yield, when distilled, burning and lubricating oils, paraffin wax used for candle making, and ammonia in quantities sufficient to make the shale industry a profitable one. The industry is carried on to a large extent in Scotland. About two million tons of oil shale are annually worked in the United Kingdom; the quantity in 1891 was 2,352,471 tons; in 1911, 3,116,803 tons; and in 1919, 2,759,165 tons.

GEOLOGICAL TERMS OFTEN USED IN MINING

Dip of Strata.—Beds are rarely found level or horizontal, but are generally inclined at various angles between the horizontal and the vertical. The inclination or slope of a bed is termed *dip*, and is measured by the angle which the plane of the bed makes with the plane of horizon (see fig. 7). The amount of dip may be stated in degrees or in inches per yard. In mines it is usually stated in inches per yard; thus a seam may be said to dip 1, 2, 3, or more inches as the case may be in each yard of distance. It is necessary to note the direction as well as the amount of dip; for example, we may say a seam dips 5° *south-east*. The angle of dip may be arrived at by using an instrument called a *clinometer*, or by an ordinary levelling instrument. When it is required in mines to drift at a certain inclination, the clinometer may be used to keep the drift at the proper angle, but more commonly a wood level with a centre plummet is used. Miners working a seam in the opposite direction to the dip say they are working to the *rise*. On geological maps, which are coloured to indicate the rocks that crop out at the surface, and on some mining maps, the

amount and direction of the dip of the beds are indicated by a small arrow pointing in the direction of the dip, and a figure placed alongside, thus "↓ 5°". When strata are vertical the dip is 90°.

Strike.—Geologically the term *strike* is used to denote the line of any bed at the surface measured at right

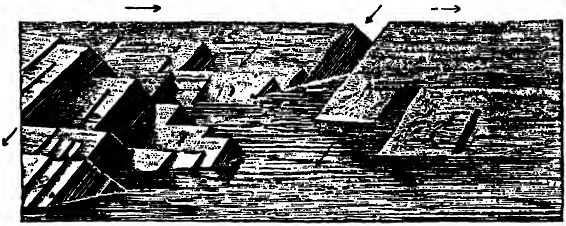


Fig 7.—Dip and Strike. The horizontal arrows indicate the Strike; those pointing to the left-hand corner show the Dip.

angles to the line of dip. Thus if the strata dip due south, the strike will be east and west, as it is at right angles to the dip and is therefore level. The dip of any bed is the line of its greatest inclination; the strike of any bed is at right angles to that line, and is therefore the line of no inclination. (See fig. 7.)

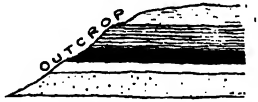


Fig. 8.—Outcrop of Horizontal Strata

Outcrop or Basset Edge.—Very often students have difficulty in distinguishing between strike and outcrop, the two terms being frequently said to mean the same thing. This, however, is a mistake. The outcrop of a seam of coal or bed is where it terminates by coming to the surface, or where it crops out by "coming to the day". If a seam crops out on the slope of a valley, the line of outcrop may very often be traced along the valley for some distance. The difference between strike and outcrop is—*the former*

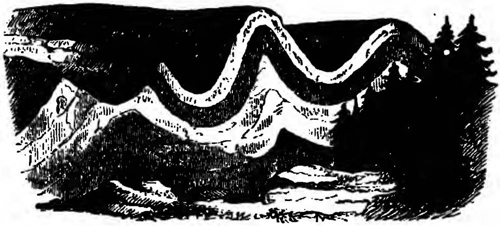


Fig. 9.—Section across the Jura Mountains, showing three Anticlines and two Synclines. The crest of one Anticline has suffered denudation

is always at right angles to the dip of the beds, the latter is the line of the beds where they appear at the surface and is rarely at right angles to the dip, as it changes its direction with the variations of the surface, and so generally forms a wavy band. Strike and outcrop are the same when the surface is level, or when the bed dips at 90°. Strata which lie horizontally have neither strike nor dip, but may have outcrop (see fig. 8).

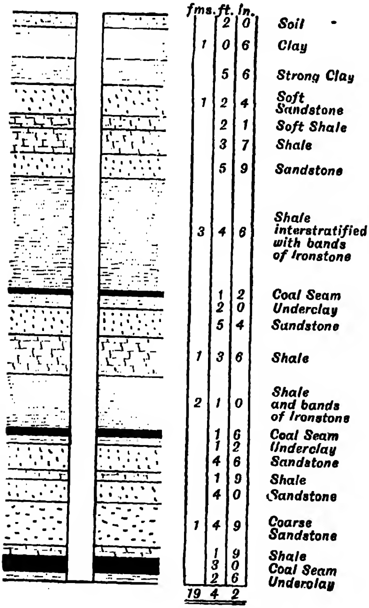


Fig. 10.—Section of Coal Shaft

Anticlinal.—When strata dip away from a central line or axis towards a hollow on each

side, the ridge is termed an anticlinal axis or anticlinal curve, or anticlinal.

Synclinal.—When strata rise on each side from a central line or hollow, they form a syncline, the hollow being termed a synclinal curve or synclinal axis (see fig. 9).

Section of Strata.—When beds are cut through in a line more or less perpendicular to the line of stratification, we obtain what is termed a "section". Thus we may obtain sections of strata in quarries or railway cuttings. The best sections are got in sinking shafts or in putting down bore-holes. The exact thickness and character of each bed may be noted for hundreds of feet in depth and shown in their relative position as a section by the draughtsman. (See fig. 10.)

CHAPTER II

COAL: VARIETIES, COMPOSITION, AND MODE OF OCCURRENCE

Varieties of Coal.—Coal varies so much in chemical composition and physical conditions that it is necessary to classify the various kinds for scientific and general purposes. Since the change in coal as it passes from one variety to another is gradual, it is difficult to draw a line of demarcation between the several varieties. As we have already said, coal is of vegetable origin; and the amount of chemical change that has taken place in the conversion of the vegetable matter into coal forms a basis for classifying the different varieties of coal. Another basis is to be found in the colour, weight, and heating power, and a classification based on these qualities answers well for general purposes.

The process of conversion of coal from vegetable matter is a gradual one, as is shown by the following

table, which illustrates the stages of the change from wood to anthracite. To render this clear, the amount of carbon is taken as constant, and the relative proportions of the other constituents increased in the same ratio. When this is done, the proportions of hydrogen and oxygen present in the various classes are readily apparent.

	Carbon.	Hydrogen.	Oxygen and Nitrogen.
Wood (mean of several analyses) ..	100	12·18	83·07
Peat " " "	100	9·85	55·67
Lignite " " "	100	8·37	42·42
Ten-yard Coal from South Staffordshire	100	6·12	21·23
Steam Coal from Northumberland ..	100	5·91	18·32
Steam Coal from South Wales ..	100	4·75	5·28
Anthracite, Pennsylvania, U.S.	100	2·84	1·74

—*Percy.*

Peat.—This comes first in the classification, as it is believed to represent the first stage in the conversion of vegetable matter into coal. Peat is generally found in bogs, swamps, and marshes, in beds varying from 1 foot to 40 feet in thickness, and contains a large percentage of water. The chemical composition is variable, but the following is the analysis of a typical sample:

Peat	Moisture	29·3 per cent.	Oxygen	17·5 per cent.
	Carbon	42 "	Nitrogen	1·7 "
	Hydrogen	5·1 "	Sulphur	6 "

Ash, 3·8 per cent.

A large part of the surface of Ireland is covered with peat-bogs, from which peat is dug and used for house fuel.

Lignite or Brown Coal.—This is vegetable matter incompletely mineralized. It is found in formations more recent than the Carboniferous; for example, it is worked in Germany from the Trias. It has a woody texture, varies in colour from brown to black, burns with much smoke, and emits a sickly odour. It has small heating power, and yields a large amount of ash. The following is the analysis of a typical sample:

Lignite	Carbon	58·78 per cent.	Nitrogen	·15 per cent.
	Hydrogen	4·04 "	Ash	5·94 "
	Oxygen	20·8 "	Moisture	10·28 "

This variety of coal is not found in large quantities in Great Britain, but on the Continent there are large deposits of it.

Gas-coal.—Coals which contain a considerable quantity of "disposable" hydrogen are used for making gas, and are called gas-coals. Coals yielding about 10,500 cubic feet of gas per ton are considered good. They should not contain more than 1 per cent of sulphur or 6 per cent of ash. *Cannel-coal* is the best example of gas-coal. It is a very compact coal of black or brownish colour. It does not soil the fingers, and can be cut and shaped into ornaments. It kindles readily, and burns freely and quickly. It is specially valuable for making gas, as it contains a large quantity of hydrogen. The following is the analysis of a typical sample:

Cannel	Carbon	80 per cent.		Nitrogen	1·5 per cent.
	Hydrogen	6 ,,		Sulphur	1·0 ,,
	Oxygen	7 ,,		Ash	4·5 ,,

The Tyne Boghead Cannel, Northumberland, yields about 13,155 cubic feet of gas per ton. This is very rich cannel.

Bituminous Coal.—From the lignite or brown coal we pass by almost imperceptible gradations to the bituminous or true coal. It is chiefly found in the coal measures, and includes *house-coal*, *steam-coal*, and *caking-coal*. The best house-coals are black and glossy, ignite easily, burn without much smoke, give off great heat, and leave very little ash. The steam-coals are dull black, ignite rather slowly, give off little smoke or flame, but intense heat. They do not fuse and cake together, and hence are sometimes termed free-burning. The best steam-coal is yielded by the South Wales basin. The caking-coals are chiefly used for making coke. When burning, they fuse and cake together; and when burnt in a closed vessel, and the liquid and gaseous products driven off, the residue contains in the form of coke a relatively increased percentage of carbon. The principal caking-

coals are found in the West Durham district. A typical analysis of a bituminous caking-coal is as follows:

Bituminous	Carbon	82·56 per cent.	Nitrogen	1·65 per cent.
	Hydrogen	5·36 ,,	Sulphur	0·75 ,,
	Oxygen	8·22 ,,	Ash	1·46 ,,

Anthracite is supposed to be the last stage of the change of vegetable matter into coal. It is jet black, and has a brilliant lustre. It is difficult to ignite, gives off intense heat with little or no smoke, and is denser, harder, and more brittle than bituminous coal. The following is a typical analysis:—

Anthracite	Carbon	91·5 per cent.	Nitrogen	1·0 per cent.
	Hydrogen	3·0 ,,	Sulphur	1·0 ,,
	Oxygen	1·5 ,,	Ash	2·0 ,,

Anthracite is found in South Wales and Ireland in comparatively small quantities. In America there are immense deposits, which are extensively worked.

Graphite, plumbago, or *black-lead* is believed to be an extreme kind of anthracite, farther removed from the vegetable origin. In general it may be looked upon as a kind of coal consisting of almost pure carbon, the volatile elements having been almost completely withdrawn. As it occurs chiefly in the primary formation or associated with igneous rocks, geologists are undecided as to its origin.

Uses of Coal.—Coal being of so many varieties admits of being applied to various purposes. Some kinds of coal are better adapted for certain purposes than others. The suitability of coal for any particular purpose depends upon its *chemical composition*, its *behaviour whilst burning*, its *specific gravity*, its *hardness or softness*, and the *amount* and *temperature of the heat* which it gives off during burning.

All coals contain more or less sulphur, and more than a certain proportion of this ingredient renders coal unfit for particular purposes; for example, in steam-coals it

causes clinkers; in gas-coal it causes strong odour and vitiates the atmosphere when the gas is burnt.

The quantity of moisture, the amount and quality of the ash left after burning, and the character of the flame often determine the fitness or unfitness of coal for certain uses.

Moisture in coal reduces the calorific value of the coal, as a certain amount of the heat is expended in driving off the moisture.

The smaller the proportion of ash, the more valuable the coal for whatever purpose it is used, because, being incombustible, the ash may be regarded as an adulterant.

Coal for metallurgical purposes must be hard, should contain very little sulphur, yield intense heat with little flame, have no tendency to cake, and leave a small amount of ash.

Coal for coking should be highly bituminous, and contain only a small proportion of sulphur and ash. It should possess the property of agglutination, or caking, when heated. Washing of the coal is frequently resorted to in order to reduce the amount of sulphur and ash; and to produce coke in strong columnar masses the coal is frequently crushed to a uniform small size. The composition of Durham coke varies within the following limits:

	Per Cent.
Carbon	85 to 95
Sulphur	·5 to 2·0
Ash	4·0 to 12·0

Coal for steam purposes should be free from much sulphur; yield a large amount of heat of high evaporative power; should be hard, so as to avoid loss by breakage—or the "small" produced will run through the bars; and should not cake too much.

Coal for household purposes should be clean; leave very little ash; burn bright and cheerful with little smoke; contain few impurities; and be of a rather hard nature, so as to produce a large percentage of "round". Some

caking-coals are well adapted for household uses, as they form a cake over the top, which causes the heat to be thrown into the room.

Coal for gas-making.—The chief requirement for this purpose is that the coal should yield a large proportion of hydrogen. Cannel yields the largest quantity and the best quality of illuminating gas, but bituminous coals are now also much employed in gas-making, as calorific value is considered of more importance than candle-power.

Mode of Occurrence of Coal.—Coal occurs in the stratified rocks, and is found in beds or seams parallel to the stratification of the rock beds above and below. It is interstratified with beds

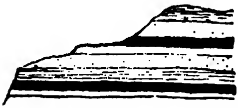


Fig. 11.—Coal Measures occurring nearly horizontal

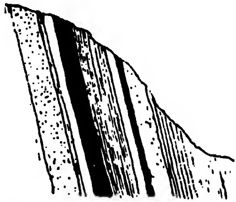


Fig. 12.—Coal Measures occurring nearly vertical

of shale, sandstone, and clay ironstone. Fortunately the coal measures as a whole in this country lie more or less horizontally, or, at any rate, at a slight angle of inclination; sometimes, however, they occur at a high angle of inclination, and more rarely still almost vertical (see figs. 11, 12). Seams of coal that are found at a high angle were not originally so; they, no doubt, were deposited on a more or less horizontal surface, but owing to subsequent upheavals and depressions they have become bent and folded, and in some instances have been thrown almost in a vertical line.

A seam of coal rarely consists of pure coal from the top to the bottom. It is frequently made up of two or more layers varying in composition, hardness, and thickness. One part of a seam may be of excellent quality

and another part of inferior quality; one part may be suitable for one purpose, say for house-coal, and another may be better adapted for steam purposes.

A seam may be split up into two or more sections by bands of black shale, fire-clay, &c. These usually interfere very much with the working of coal-seams, and pieces are liable to become mixed with the coals, and thus materially increase the labour of cleaning the coals to render them marketable when they reach the surface. Occasionally pieces of shale and other impurities occur interspersed and embedded in the coal-seam, a frequent impurity being *iron pyrites*, or bisulphide of iron. This is known to miners as "brass", and seams containing much of it are said to be "brassy". It is frequently found in seams in small pieces, but sometimes also in thin layers. It requires careful picking out, as its presence renders coal unfit for many purposes. It is also dangerous if mixed with coal in heaps, as it is liable to produce spontaneous combustion.

An example of a seam split up into sections of different composition is the Maudline Seam of the Durham District:

	Feet.	Inches.
Good Coal	1	6
Clay Band	0	1½
Good Coal	1	6
Splint Coal	0	2½
Good Coal	1	2
Thickness of Seam ..	4	6

Fig. 13

Another example is the Silkstone Seam of the Yorkshire Coal-field:

	Feet.	Inches.
Inferior Coal	0	8
Good Coal	1	9
Dirt	0	6
Good Coal	2	7
Thickness of Seam ..	5	6

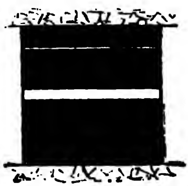


Fig. 14

Seams seldom continue of the same character over extensive areas. They frequently become changed in thickness, quality, hardness, and composition. Clay bands may appear, thicken, or disappear.

Seams vary in physical conditions; some are soft and tender, easy to work, and produce much dust and small coal. The latter reduces the value of the seam unless it is a caking-coal. Other seams are very hard, and the coal so difficult to get as to require blasting. These seams produce large blocks, known as " round " coal, and very little small coal. Thick seams may have hard and soft coal in their section; in such cases the soft is generally worked first to assist in getting the hard.

Cleat of the Coal.—Nearly all rocks have joints or cracks which traverse them in two directions, generally at right angles to each other, and divide them into blocks of greater or less dimensions. They may be seen in any quarry or mine, and assist greatly in blasting operations. Coal also is a *jointed* rock, and the joints are of as great importance in mining as in quarrying. The vertical thickness of coal is made up of layers or thin laminæ, which run parallel to the rock beds. At right angles to these lines of lamination are two sets of joints at right angles to each other, one of which is very clear and distinct, producing a smooth vertical section, or " face " of coal, to which the name " slyne ", or " plane " is applied by miners. The other set at right angles to the " plane " is less well marked and is called, by the miner, the " ends ". The direction of this cleat is sometimes continuous and regular for long distances. In some seams the cleat is very indistinct, perhaps more so in hard than in soft seams.

The structure of cleat is very useful in mining, as it assists the working of the coal, and the direction of the working places is frequently determined and guided by it.

CHAPTER III

FAULTS, HEAVES, SLIPS, THROWS, TROUBLES, AND OTHER INTERRUPTIONS MET WITH IN COAL MINES

Faults are frequently encountered in the working of coal-seams, and are well known to all miners. A fault may be defined as a crack or fissure in the strata, more or less vertical, accompanied with a vertical displacement of the beds either upwards or downwards from their original position.

Originally, as previously stated, the beds were deposited in a continuous horizontal line, but owing to various movements of the earth's crust in former periods they have been fractured, and portions have been elevated and depressed in such a way that, instead of in continuous beds, we find them detached and lying at different levels. It is not uncommon, in working a coal-seam, to find it suddenly terminate and stone appear in the place of coal, accompanied with indications of a fissure filled with a solid mass of broken stone extending upwards and downwards. This is a fault, and the continuation of the coal-seam will be found at a different level either above or below. This is illustrated in fig. 15.

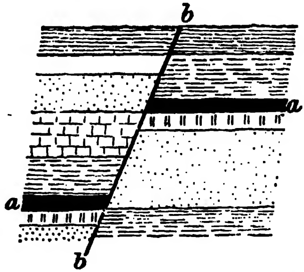


Fig. 15.—Section of Faulted Beds. The coal-seam *a*, *a*, and beds on each side of the fault, were continuous. The fault *b*, *b* hades or inclines to the left of the diagram at angle of over 60°.

The width of such a fracture or fissure varies from a fraction of an inch to several feet. It is usually filled up completely from side to side of the fault, the in-filling

material frequently consisting of broken pieces from the rocks through which the fault passes; in some cases the fissures contain valuable minerals, which may have been deposited chemically from below, or from rocks adjacent to the fault. One fault at a colliery in the county of Durham, when drifted through, was found to be eight feet in width, and filled from side to side with pure white barium sulphate. The fissures which contain ore are seldom rich enough in coal-mining districts to be workable.

The *amount of vertical displacement* of faulted beds varies from an inch or less to thousands of feet. When the displacement is not more than a few feet the fault is generally known to miners by the terms *hitch*, *heave*, and *slip*, but when the amount may be stated in fathoms it is frequently termed a *trouble*. The *amount of the displacement* is called the *throw*, and is the perpendicular distance from the *thill* or floor of a bed or of a coal-seam on the elevated or rise side of the fault to the thill or floor of the same bed or coal-seam on the depressed or dip side of the fault or vice versa. (See fig. 16.)

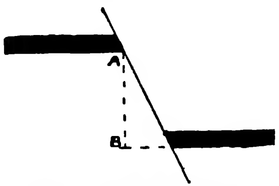


Fig. 15.—Measurement of the vertical displacement or "throw" of a Fault. The dotted line A B is line measured, not the slope of the fault.

A fault is termed a *downcast*, *downthrow*, and *dipper*, or an *upcast*, *upthrow*, and *riser*, according to the side from which it is approached. For example, if a drift is being driven in a coal-seam in a south-east direction, and a fault is met with which displaces and throws the seam down a depth of 20 feet below the drift, the fault would be termed a *dipper* or *downthrow* south-east of 20 feet. If the drift had approached the same fault on the downthrow side in a north-west direction, it would have been termed a *riser* or *upthrow* north-west of 20 feet.

Hade of a Fault and its Use in Mining.—The

line of fracture is seldom exactly vertical or at right angles to the dip of the beds. It varies to some extent according to the nature of the strata through which it passes, being more regular and nearer the vertical line in hard than in soft beds.

The *inclination of a fault* is measured from the vertical line in degrees; or, in other words, it is the angle formed by the plane of the fault and the vertical plane. The inclination is termed the *hade* of the fault. *Hade* is applied to the inclination of a fault from the perpendicular, and the term *dip* is applied to the inclination of strata with the plane of the horizon. *Hade* is measured

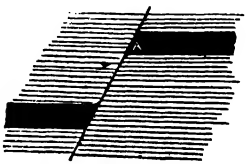


Fig. 17.—The Hade at A indicates that the Fault is a "dipper"

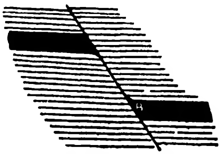


Fig. 18.—The Hade at B indicates that the Fault is a "riser"

by the angle which a fault makes with the perpendicular plane; *dip* is measured by the angle which a bed makes with the horizontal plane.

The hade of a fault is of great value in mining in assisting to determine whether the throw of the beds is upwards or downwards, or, in other words, whether the fault is a riser or dipper. The hade is regarded as usually affording a good indication of the vertical direction in which the beds are to be searched for on the unknown side of the fault. The rule used by miners is: the fault hades or slopes in the direction of the downthrow side—that is, the fault leans towards the observer at the top and from him at the bottom when it is a downthrow. In other words, if the angle made between the thill of a coal-seam and the fault is an acute angle, as at A in fig. 17, the fault will be a dipper. On the contrary, if a fault leans or slopes towards the observer at the bottom,

and from him at the top, or forms an acute angle with the roof of the coal-seam, as at B in fig. 18, the fault will be a riser. These rules only refer to the direction, and not to the amount of throw of the beds.

Other indications of the direction of the throw usually accompany faults, which are looked for when searching for the faulted portion of a seam. One of these is often known as the tailing up or down of the coal in the fissure in the direction of the displaced seam; this is usually regarded as a good indication. Another, but one which is not so reliable as the foregoing, is, that a seam is usually found to rise for a short distance to a fault which is a dipper and dip for a short distance to a fault which is a riser. A reference to figs. 17 and 18 will show that the same fault may be a riser or a dipper according to the side from which it is approached.

In a mine, when it is necessary to find the faulted portion of the coal-seam, or, as it is commonly termed, *to prove the fault*, the above-mentioned indications as to the direction of the displacement are first observed, and the excavations are determined by them. If the coal is not found by following the slope of the fault either up or down, as the case may be, this is discontinued, and a drift is driven straight through the fault into the regular strata beyond. The beds are then examined at the face of the drift and compared with an accurate section of the strata above and below the seam, taken in some part of the mine, say the shaft section, to find the position of the faulted beds and the amount of the throw. This comparison is sometimes very difficult to make, owing to the character of the beds varying so little, and especially with thick beds. If the position of the faulted beds is still undetermined, a bore-hole may be tried next, either upwards or downwards or both, near the face of the drift. This will give a correct vertical section to compare with the shaft section. Sometimes the bore-hole is continued until it bores into the coal-seam, thus accurately proving the position of the faulted portion.

The Course of Faults.—Some faults vary considerably in their course and also in the amount of throw; others run in a more or less straight course for long distances, and vary but little in the amount of displacement. The great Pennine Fault, which commences in the south of Scotland and extends into Derbyshire, can be traced for a distance of 130 miles. It dislocates the strata to an extent varying from 3000 to 6000 feet. Another example of a large fault is illustrated in fig. 19.

Faults do not usually terminate abruptly, but die out

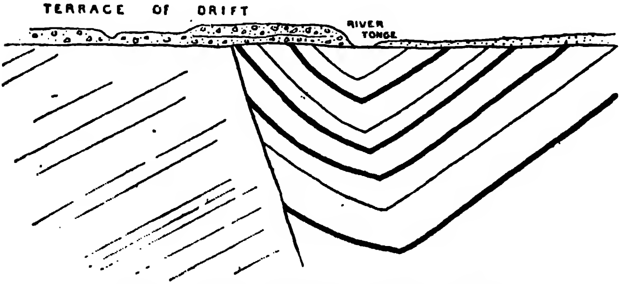


Fig. 19.—Section across the Irwell Valley Fault

by the amount of the throw becoming gradually less, and in some cases by splitting up into numerous minor branches, which gradually thin out.

The miner usually finds that most large faults throw off smaller shoots or branches. These offshoots generally run in several different directions, and commonly show smaller displacements than the main fault. Mines which have large faults running through them are usually very much hampered in the workings by these lesser faults.

Varieties of Faults.—Faults have been classified as follows:

1. *Ordinary or Normal Fault*: when the beds are displaced and the faulted portion can be discovered by the rules given above. Such faults occur in large numbers,

and are to be found of more or less throw in almost every colliery. An example is given in fig. 20.

2. *Step Fault*: one fault after another in almost regular order, so that the faulted beds have the appearance of

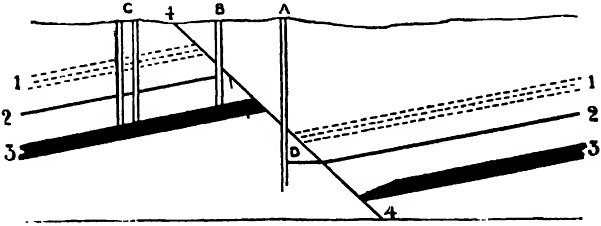


Fig. 20.—Diagram of the Corbyn's Hall Fault, South Staffordshire. (After Juke's Geological Survey.)

A, B, C, Colliery shafts; D, Gate road. 1, Beds containing water. 2, Brooch coal. 3, Thick coal. 4, The fault.

being a series of steps. This often occurs with small faults, but the total amount of throw may sometimes be large. An illustration of a step fault is shown in fig. 21.

3. *Trough Faults*: when the beds are thrown down between two faults which are a short distance apart.

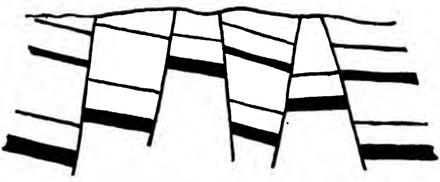


Fig. 21.—Section of Coal-seams dislocated by Step Faults. The seams were originally continuous.

The example given in fig. 22 occurs in a Northumberland colliery. The beds are thrown down a depth of 18 feet between two faults, which are about 300 yards apart, giving the faulted beds the appearance, in vertical section, of a huge wedge. Inverted trough faults also occur.

4. *Reversed or Overlap Fault*: when the beds are displaced in such a manner that the dip-side beds are found under the rise-side ones. This is the very opposite of the ordinary fault, and to find the faulted seam the rules must be reversed.

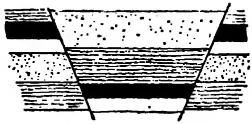


Fig. 22.—Trough Fault, in which the beds are thrown down 18 ft.

Reversed faults are rarely met with in mines; when they do occur they are generally a source of great trouble and expense. When the fault is encountered, and it is not known to be a reversed one, the rules used to find an ordinary fault are first employed; the opposite course is not attempted until there is good reason to conclude the fault to be a reversed one. The example given in fig. 23 occurred in the writer's experience, and was proved after a bore-hole had been put both upwards and downwards, the latter bore-hole passing through the seam 9 feet down. Reversed faults are much more common in formations older than the coal measures, particularly those which have undergone a great deal of contortion, and where thrust-planes are common.

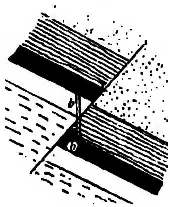


Fig. 23.—Reversed Fault. *b b*, Bore-hole put down 9 ft. and the coal-seam found.

Parallelism of Faults.—It has frequently been observed that large faults run more or less parallel to each other across coal-fields, and that they can be divided into two sets, one set having a tendency to run parallel to the strike, while those of the other set are in the line of the dip and at right angles to the former. For instance, in the coal-field of Northumberland and Durham there are one set of faults running roughly parallel to each other in an east-and-west direction, and another set running in a north-east and south-west course.

General Appearance of Faults.—The cheeks, sides, or walls of faults are frequently grooved, smoothed, and glazed, as though they had been rubbed roughly against each other and partly fused. This appearance is termed *slickenside*. The same kind of thing may be often seen in joints and jacks near large faults.

The coal adjacent to faults on both sides is in nearly every case more or less altered in thickness, quality, and dip. Sometimes the seam is increased in thickness, but more often the opposite occurs. The coal is usually more or less deteriorated in quality, so much so in some instances as to be unmarketable; this is usually termed *hitch coal*. In some cases the coal is rendered very much harder, in other cases softer, than the normal, and occasionally no change of any kind is observed in working a seam near a fault until the fault is reached. More frequently changes are noted, particularly of dip; the seam as previously mentioned often dips to a riser and rises to a dipper fault, and in some cases the strata are bent, tilted, and contorted a great deal.

Dykes.—These dislocations met with in coal-mines rank next in importance to faults. The term dyke is frequently misapplied; for instance, the miners in the Northumberland and Durham Coal-field use it as a synonym for fault, thus the " Ninety-Fathom Fault ", which runs from Cullercoats to Whittonstall, via the north-west side of Newcastle-upon-Tyne, is always termed the " Ninety-Fathom Dyke ".

A dyke is a fissure in the earth's crust filled with igneous rock, which has been forced upwards in a molten condition and has since cooled and hardened.

Dykes descend to unknown depths; but they sometimes terminate upwards before reaching the surface by spreading horizontally or laterally among the rocks. Sometimes when the molten matter on reaching the surface has overflowed laterally, and thus formed a more or less horizontal igneous mass, it has afterwards become overlaid by stratified rocks.

The igneous rock forming a dyke is usually much harder and more difficult to denude than the strata through which such dykes cut, hence it is a very common occurrence for a dyke to be left protruding above the level of the surrounding country, like a long wall, owing to the more rapid denudation of the sedimentary rocks. (See fig. 24.)

Dykes vary much in length: some can be traced for short distances only, others extend longitudinally for many miles. They also vary in width from less than a foot to many yards. They generally have very little hade, being nearly vertical, with their sides usually parallel and very smooth. In many cases they are unaccompanied by any displacement or throw of the strata through which they pass, but sometimes they constitute both dyke and fault; in most cases, however, the ends of the strata adjacent to the dyke are upturned and contorted.

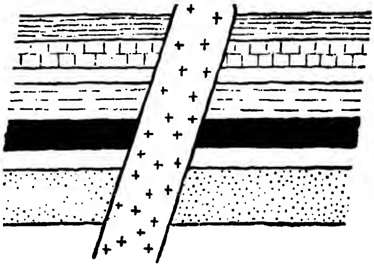


Fig. 24.—Dyke of Igneous Rock (Basalt) cutting through Coal-seam and Beds of Sandstone, Shale, &c.

For some distance on each side of the dyke the beds are found to have undergone considerable alteration or metamorphosis by the intense heat of the molten lavas which have been ejected into the fissure from the interior of the earth. Sandstones are hardened and changed to quartzite; shales are hardened and rendered slaty; limestones are altered to marble; and coal is often so changed as to resemble coke or inferior anthracite, and sometimes it is burnt to cinders and soot.

In the south-west district of the county of Durham the

Cockfield Dyke is found extending in a north-westerly course for several miles. It has been extensively quarried at various places along its course for road metal. It varies in width, but averages about twenty yards, with thirty or forty yards of bad coal and cinder on each side in every seam it cuts through. This has been proved by the many drifts which have been put through it in different mines.

There are some dykes the in-fillings of which consist of debris resulting from the denudation of sedimentary rocks, and carried in from the surface. Such dykes are not numerous, and are sometimes termed *soft dykes*, in contradistinction to the hard or igneous dykes. The latter are sometimes termed *whin dykes*, because their contents consist of basalt, which is the whin rock of the quarryman.

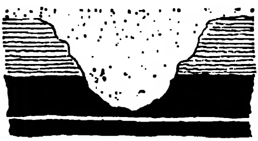


Fig. 25.—Stone Balk in Coal-seam

Balks.—These are another kind of interruption to the regular continuity of coal-seams. They are sudden depressions of the roof of the coal-seam, whereby the latter is considerably reduced in thickness. They occur often with a sandstone and sometimes with a shale roof.

Balks are probably due to denudation, the channel or wash resulting from the denudation having been filled in when the bed now forming the roof of the coal-seam was deposited. An illustration of a balk is given in fig. 25. In the example the shale resting upon the coal-seam is 5 feet thick and has been washed out for a width of 6 feet, and the coal-seam has been denuded nearly down to the thill. The bed above the shale consists of sandstone, and the balk is filled in with the same rock.

Balks vary much in dimensions—in length, breadth, and depth. Sometimes the whole thickness of the coal-seam has been washed out, in others only a slight depression occurs. Balks are features of small importance

geologically, but in mines where they are numerous they are a source of trouble and expense.

Rolls.—These are the reverse of balks; the thill or floor of the seam suddenly rises until it may cut through the coal and meet the roof (see fig. 26). They occur in nearly all coal-mines, and are probably due to the undulations of the old land-surface upon which the coal was deposited and which now forms the thill of the coal-seams.

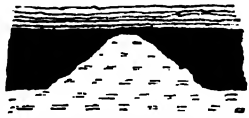


Fig. 26.—Example of a "Roll" occurring in a Coal-seam

Nip-outs.—These occur when the roof and thill approach each other, and diminish the usual thickness of the coal-seam or *nip* it out completely. They are not so frequent as balks and rolls.

Swellies.—These are thickenings of the coal-seam for a short distance and vary much in dimensions. As we have said before, the old land-surfaces, upon which the vegetation grew which now forms coal-seams, would probably be more or less uneven, and the swellies found in working coal-mines are merely the depressions or hollows which then existed (see fig. 27).

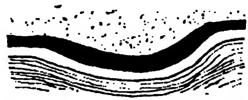


Fig. 27.—Example of a "Swelly", in which the coal is thickened from 3 feet to 6 feet

Washes or Wash-outs.—Sometimes a seam will terminate abruptly by coming against sand and gravel which has the appearance of once having been the bed of an ancient river (see fig. 28). The whole of the beds and seams from the surface to the depth attained by the river must have been denuded, and a valley formed of varying width and extending the length of the river. The valley would subsequently become filled up, probably with drift sands, gravels, and clays of glacial age.

A remarkable wash occurs in the county of Durham.

It is known to extend from Durham along the Team Valley to the Tyne, and has been well proved along its course by the workings of various collieries.

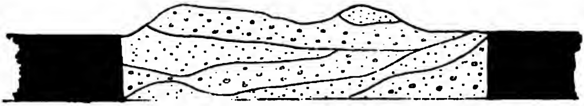


Fig. 28.—Example of Wash-out filled with Sand or Gravel, as proved by a drift from seam to seam

Other Irregularities in Coal-seams.—Other irregularities and interruptions encountered in the working of mines might be mentioned, such as the splitting-up of seams into two or more portions, whereby a workable seam may become unworkable owing to the bands of stone which divide it into thin layers of coal—too thin to be profitably worked. Then there is the reverse occurrence of two distinct seams running together in some places to form one thick seam. Then there is also the thinning or dying out of seams.

CHAPTER IV

SEARCHING AND EXPLORING FOR COAL-SEAMS

Prospecting.—Before expending money in boring or sinking a shaft, it is necessary to have some evidence of the existence of coal-bearing strata. A geological survey will be first required if the locality is an unexplored one, to determine the existence and position of the Carboniferous Formation, and here a knowledge of geology becomes valuable. If this Formation does not exist in the locality, further endeavours to find coal will most likely prove futile and a waste of money.

When the Carboniferous Formation is discovered, the next step is to ascertain if it contains the coal measures—their extent and position; and, if possible, to find the outcrop of a seam of coal. The sides of valleys and ravines, the banks of rivers and streams, wells, quarries, and cuttings should all be minutely examined for indications of outcrop of shale, fire-clay, or coal. The beds of streams may afford a valuable means of information: pieces of coal and other mineralogical specimens may have become detached from the outcrop of beds at points higher up the stream; the course of the latter should then be followed, and the sides carefully examined for indications of the outcrop.

The actual outcrop frequently becomes overlaid and hidden by newer deposits; when this is the case it may be found by the darker appearance of the soil, due to the admixture of small pieces of coal and shale, which will be detected by a careful examination. Sometimes the existence of a coal-seam has been revealed by very dark soil being overturned in ploughing, or by spring water, which has filtered through a coal-seam containing iron pyrites, forming a deposit of iron at the mouth of the spring.

Shale may be said to be a very good indication of the existence of coal in the vicinity, but not one to be too much depended upon, unless, accompanying the shale, such corroborating evidence as is afforded by the impressions of ferns and other vegetable remains and animal fossils characteristic of the coal measures is found. Some of the shales found in the Silurian very closely resemble those of the Carboniferous in colour, &c., and unless some characteristic fossil remains are found to disclose the formation to which they belong, shale may mislead. When an actual outcrop of a seam of coal is not discovered, the prospecting should be very carefully conducted to find the other indications, and, if possible, to find some coal-measure fossils before commencing to bore for coal.

The Workable Depth of Coal-seams.—The existence of the coal measures may have been ascertained by a geological survey of the locality, but the coal-seams of a workable thickness may occur at too great a depth to be regarded as a profitable venture. There are various circumstances to be considered in deciding the question of the depth at which the working of coal ceases to be practicable and profitable.

1. *Temperature of the Rocks.*—The generally accepted limit of depth at which coal can be worked is 4000 feet below the surface, although up to the present no coal shaft in this or other countries has been sunk to that depth. One reason for this limitation is the observed gradual increase of temperature as we descend into the crust of the earth. On this point careful observations in various parts of the world, of borings, sinkings, and tunnels, have been made over a number of years, and results obtained which may be relied upon.

Below the surface, to a certain point, the temperature of the rocks is influenced by the temperature of the atmosphere. This point is from 60 to 90 feet below the surface, and an imaginary line drawn at this depth is called the line of constant temperature, while the part overlying this imaginary line is known as the zone of variable temperature. Deeper than this the temperature of the rocks increases with the depth. The increase is not uniform in amount at all places; it varies at different depths in the same bore-hole, in different strata at the same depth, and in different localities, as shown by the following:

	Temperature Noted. Deg. F.	Depth of Observations. Feet	Rate per 1 deg. F. Feet.
St. Gothard Tunnel	87	5578	82
Mont Cenis Tunnel	85	5280	79
Ashton Moss Colliery, Manchester	85¼	2790	77
Wearmouth Colliery, Durham	71¼	1584	70
Rosebridge Colliery, Wigan	94	2445	54
Bolden Colliery, Durham ..	79	1514	49

The average rate of increase may be taken to be about 1° F. for every 60 feet of descent. At a depth of 4000 feet the increase of temperature would amount to 66°; and 50°, which is the uniform temperature over England at 50 feet below the surface, added to this, gives a temperature of 116°.

This, of course, in mines would be quickly reduced near the shafts by the rapid circulation of large quantities of air, and would possibly be further reduced by other means.

The temperature at any depth may be readily obtained by the following formula based on the foregoing data.

Let t = temperature in degrees Fahrenheit.
D = depth of shaft or bore-hole in feet or depth at which it is desired to find the approximate temperature.

Then
$$t = 50 + \frac{D - 50}{60}.$$

Example.—Find the temperature in degrees Fahrenheit at a depth of 2500 feet.

$$t = 50 + \frac{2500 - 50}{60}$$
$$= 91° \text{ nearly.}$$

The Coal Commission of 1866–71 considered the question of temperature in deep mines, and it was further investigated by the Commission of 1901. The former Commission concluded that the blood-heat limit (98°) was the highest temperature at which men could do manual labour in mines. This limit is reached at a depth of 3000 feet; but it is generally believed that in mines at 4000 feet the temperature could be artificially reduced to the workable limit. At the Produits coal-mine, Flenn, Mons, Belgium, the workings have reached a depth of 3773 feet, and at Pendleton Colliery a depth of 3483 feet.

2. *The Thickness of Seams.*—The limit of thickness for workable seams of coal used by the Commissioners in estimating the probable quantity of coal contained in the

coal-fields of the United Kingdom was 12 inches. This is a very low limit, and even at moderate depths the conditions for working a seam of only 12 inches would have to be exceptionally favourable, such as the existence of a bed of good fire-clay forming the floor of the coal-seam, and which would be worked and sent out of the mine for brick-making. It is only under favourable conditions that seams under 18 inches can be worked at a profit.

3. *The Mechanical Difficulties.*—At a certain depth the weight of the rope hanging from the top of the shaft will be equal to that of the safe working load, and it will thus have sufficient to safely support in itself without the additional weight of cage and coals. Taper ropes have been introduced to meet this difficulty; they are made of the greatest size at the top, and gradually taper or decrease in size to the bottom. The top end of the rope is therefore the strongest part of it, and has to bear the greatest stress, having the full weight of the rope in the shaft as well as of the cage and coal to support at the commencement of each wind. The bottom end, on the other hand, has the least stress, and does not require to be so strong. It may be pointed out that taper ropes are little used, as there is difficulty in manufacture. Another method suggested to overcome this mechanical difficulty is to wind in two or more stages, the winding in the underground shafts being done by electric winding-engines.

Moreover, all existing difficulties in connection with the working of coal are increased with increase of depth. The pressure from the superincumbent strata increases with the depth, and this means greater crush upon the coal and timber used for support. The effect of the crush is to break the coal and so reduce its commercial value. Roadways are rendered more difficult to maintain. Greater quantities of fire-damp are usually evolved in deep mines than in shallow ones, and generally the dangers of mining are increased.

The Economical Limit.—The opening out and the equipment of surface arrangements of deep mines are considerably greater than at shallow ones. The cost of a plant at a mine 3000 feet in depth will probably be double compared with 1000 feet. The cost of production and maintenance would be greater, because the working of the mine generally would be more expensive, as well as the hauling and winding charges. The working of thin seams at great depth is a problem for the future, to the elucidation of which mechanical appliances, with electricity as a motive power in place of manual labour, will no doubt largely contribute.

Deep Coal-mines.—The deepest mine which is at present worked in Great Britain is the Ashton Moss Mine, near Manchester. The lowest seam sunk to was reached at a depth of 472·5 fathoms, and the total depth of the shaft is 475 fathoms. The seams dip from the shafts, so that the depth of the workings increases as they advance.

Rosebridge Colliery, Wigan, is sunk to the Arley Mine at a depth of 403 fathoms. Harris Navigation Colliery, South Wales, is sunk to a depth of 380 fathoms. Seaham Colliery, in the north of England, is 301 fathoms to the Busty Seam.

The greatest depth that has been reached is in Belgium. Experience at the Sainte-Henriette Pit has proved that coal can be worked at a depth of 3936 feet; and when the new works are complete, the actual depth from which coal will be raised will be 3609 feet = 601½ fathoms.

The Viviers Rennis Coal Mine, at Gilly, near Charleroi, is 581·5 fathoms in depth. Deep shafts have recently been sunk at Sydney Harbour Colliery, New South Wales. Commenced in 1897, they reached the seam in December, 1901, at a depth of 2880 feet.

Outcrop Headings or Drifts.—Supposing that after careful searching and prospecting we have become

satisfied of the existence of the coal measures, the next thing is to ascertain, with as much accuracy as possible, the depth of the workable seams, their thickness and quality, dip, uniformity, the nature of the strata, and, in fact, as many particulars as can be got.

If the outcrop of a seam has been discovered, some of these particulars may be ascertained by driving headings or drifts into it for some distance along the line of the outcrop. By this means some information may be obtained of the thickness, quality, and character of the seam, and of its roof and thill. To obtain more accurate information, the drifts should be driven a sufficient distance in the direction of the dip of the strata to get clear of the outcrop coal, which is generally very much deteriorated owing to atmospheric influences.

Trial Pits.—Sometimes it is difficult to drive headings, owing to the inconvenient position of the outcrop of the coal-seam. To ascertain particulars of the seam, little shafts may be sunk to it at different points and at a short distance from the line of the outcrop—sufficient to be clear of the outcrop coal. From the bottom of each shaft the coal may be worked a little so as to see the amount and direction of dip, nature of roof and thill, quality of coal, &c.

Costeaning.—This is an old method adopted in metalliferous mining when searching for a lode of ore, and may be adopted in searching for seams lying at a high angle. From the appearance of the surface, the observer may have concluded that somewhere near there exists a vein or lode of ore, and he decides to sink two or more small shallow trial pits. Between each pair of pits a level, or drift, is driven to make a communication, and if the lode exists anywhere between them, the drift will cut through it. The same would apply if a seam of coal lying at a steep inclination came in between the two shafts.

CHAPTER V

BORING TO PROVE COAL-SEAMS

Use of Boring.—If the prospecting and the driving of outcrop headings, or the sinking of trial pits, do not give altogether satisfactory or sufficient results, or if information is required over a wider area and at greater depths than near the outcrops, the process known as *boring* is resorted to.

By boring is meant the making of a vertical hole of small diameter in the crust of the earth to ascertain the nature and thickness of the rocks, and in coal mining with the object of ascertaining the existence of seams of coal.

By the process of boring we therefore obtain:

1. Depth, thickness, character, and number of seams in a coal-field.
2. The extension or continuity over a given area of the coal-seams.
3. The amount, direction, and variations of the dip of the strata.
4. The existence of any great faults or other disturbances of the strata.

Of course, one bore-hole could not possibly give all this information; it might give correctly the depth and number of the coal-seams, but to ascertain their extension over a large area, the inclination of the strata, and the existence of any faults, quite a series of bore-holes would be required. The results thus obtained should be carefully recorded and their various positions noted, so that the requisite information may be deduced. As boring is generally done preparatory to sinking shafts, it is highly important that accurate information be obtained so as to avoid disappointment when the shafts are sunk.

The Process of Ordinary Boring.—Boring is done either by (*a*) percussive action or (*b*) rotary action, and by one of the following methods:

Percussive	1. Wood, iron, or hollow rods by hand or machinery. 2. Rope with cutting tool.
Rotary	3. Diamond drill, calyx drill, or chilled shot.

The ordinary method of boring is to use iron rods, called *bore-rods*, attached to the bottom end of which is the cutter or chisel for the actual boring of the hole. At the top is a double pair of wooden handles, called a *bracehead*, by which the rods are raised and worked in the bore-hole. To cause the chisel to cut into the rocks, two men take hold of the bracehead and raise the rods a few inches, then allow them to drop sharply to the bottom of the hole to force the chisel into the stone. To make the hole as circular as possible, at each time of raising the rods the two men at the bracehead give them a partial turn in such direction as will prevent any of the rods becoming unscrewed. This causes the chisel to drop on a fresh place each time, thus preventing it from wedging itself, and making the hole circular.

The stone at the bottom of the hole becomes broken up into very small pieces by the action of the chisel, and after working for a short time the rods are withdrawn from the bore-hole, and the chisel taken off and replaced by a cleaning-instrument termed a *sludger* or *wimble* (see fig. 35). This is lowered into the hole, and, being hollow, the debris becomes worked into it by alternately raising and lowering it; it is then raised to the surface, the chisel replaced, the rods lowered, and the boring recommenced. As the bore-hole increases in depth, additional bore-rods are screwed on at the top.

The thickness of each bed of rock is found by marking the rods when a fresh stratum is entered; the nature of the rock is determined by the contents of the sludger being carefully examined each time. This information

is duly noted in a book, so that at the finish of the bore-hole a complete section of the strata passed through has been obtained.

When the hole has descended about 50 or 60 feet, and the weight of the rods has become too great for the men at the brace-head to lift, additional appliances are required to assist them. When it is anticipated that the boring will descend to a great depth, these must be very strong and complete.

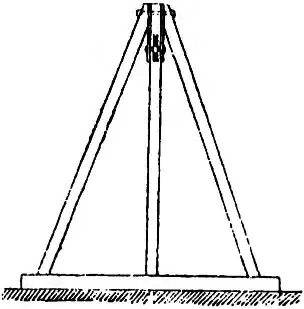


Fig. 29.—Boring Head-gear in triangular form

Description of the Principal Tools used in Boring.—For the purpose of description, ordinary boring appliances may be divided into three sections:

1. The surface erections and appliances, such as head-gear or shear-legs, windlass, spring-pole or rocking-lever, &c.

(1) *Head-gear.*—If the bore-hole is not intended to be a deep one, the head-gear may consist of three wood poles erected in a triangular form, meeting at the top, where they are bolted, and where an iron sheave is suspended perpendicularly above the bore-hole (see fig. 29). When it is expected that the bore-hole will go to a considerable depth, a similar contrivance of four wood poles, erected in a square form, is generally adopted (see fig. 30).

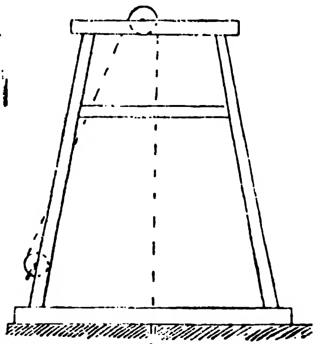


Fig. 30.—Boring Head-gear in square form suitable for deep boring

In determining the height of the head-gear, it should be remembered that the drawing out of the rods is a process which occupies much time, and the longer the lengths to be unscrewed are, the less time will be required; this of course depends upon the height of the head-gear, and therefore the higher it can be made the better. In all cases it should be arranged for the height of the head-gear to be a multiple of the lengths of which the rods are made up, so that when the rods are raised up to the top of the head-gear, a joint of them to be unscrewed will be just above the top of the bore-hole. Thus, supposing the rods are 12 feet long, the head-gear ought to be 24 feet, or 36 feet, or 48 feet.

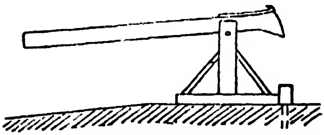


Fig. 31.—Rocking-lever used in Boring

(2) *Windlass.*—This is frequently used for raising the rods in shallow bore-holes, and may also be employed to assist the men at the bracehead in the actual process of boring. It is generally fixed in a frame, the barrel being from 12 to 16 inches diameter, and 6 feet in length.

When the bore-hole becomes deep and the windlass incapable of raising the rods, a winch may be applied. This is adapted for raising heavy weights, and is much safer than a windlass, because it is fitted up with spur-gearing, ratchet and pawl, and a brake. In very deep holes a steam winch or some other steam-engine is necessary to raise the rods.

(3) *Rocking-lever.*—When the weight of the rods becomes too much for the men at the bracehead, they may be assisted by a rocking-lever. It consists of a wood lever, 12 to 15 feet in length, provided with an iron axle working in a wood frame (see fig. 31). The distance from the top of the bore-hole to the frame forms the shorter arm of the lever, and varies in length with the depth of the bore-hole and the length of the

stroke. In boring through hard rocks, the rods are raised higher to give the blow than in soft rocks, and this requires a longer stroke of the lever. The bore-hole end of the lever is turned in the shape of a sector of a circle, so as to raise and lower the rods in a perpendicular line. Near the top of the sector is an iron hook, to which is attached a short chain from a ring on the top of the bracehead.

In working, the rocking-lever is depressed by men at the longer arm; this raises the rods in the bore-hole to the required height. The men at the bracehead give the rods a slight turn, then the descent of the rods in the hole raises the rocking-lever to its former position, ready to be again seized by the men. It is sometimes necessary to counterbalance the rocking-lever. Steam-power is advantageously substituted for men to work it when the bore-hole becomes very deep.

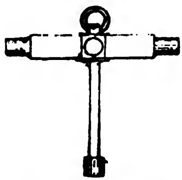


Fig. 32.—Bracehead

(4) *Guide Tube.*—This is used to keep the bore-hole as near as possible perpendicular, and great care should be taken in fixing it. It consists of a cylindrical piece of wood about 12 inches in diameter and 6 feet in length, with a hole bored through it from end to end of the same diameter as the intended bore-hole. It is securely fixed at the top of the bore-hole in a strong wooden frame, and provided with a pair of iron shutters to prevent anything falling down.

(5) *Other surface erections* usually required are, a small blacksmith shop for sharpening the chisels, &c., and sheds for men and tools; also ropes and chains for raising the rods by means of the windlass.

2. The tools required to transmit the power, viz. bracehead, bore-rods, &c.

(1) *Bracehead.*—This consists of four wood arms, about 18 inches long, at right angles to each other, and firmly secured to a piece of bore-rod about 18 inches

long, which is screwed to the top of the bore-rods when working (see fig. 32). As already explained, in the process of boring, two men opposite each other take hold of these arms to lift the rods and turn them.

(2) *Bore-rods.*—These are generally made of the best iron, and of one inch or more square section, in lengths varying from 6 to 18 feet. The square section is better than the circular and octagonal, which have been tried, because the keys can be so readily and securely applied. The rods are connected to each other by the ordinary male and female screw-joint, and the screwed part should be made of the same sectional area as the rod, for strength; the joint is thus double the section of the other part of the rod (see fig. 33). Great care should be taken in screwing up a joint to have it made tight, otherwise the threads may become stripped in working, in which event the rods below will be left in the bore-hole.

Fig. 33.—Bore-rod

The chief disadvantage of iron rods is their great weight; in a deep boring they require much power to lift them, and in the downward stroke there is a very heavy shock to the rods. To obviate this, wooden rods have been tried where the depth is great. Bore-holes as a rule become filled with water, and the wooden rods lose much of their weight in it, and are thus not subject to such enormous shocks as the iron rods. Wooden rods are generally made of pitch-pine in long lengths and large section, the latter being the chief objection to their use. Ropes have been substituted for iron rods, but have not been much employed owing to the difficulties attending their use. The bore-hole can be put down very quickly with a rope, but it is difficult to keep it vertical, and there is more liability of breakages than with iron or wood; when breakage occurs the bore-hole is entirely lost.

In the special system of boring adopted by Mather and Platt, a flat hemp rope is used with some success.

3. The tools employed in the actual boring of the hole, viz. chisels, sludgers, augers, crow's-foot, bell, and wadhook.

(1) *Chisels.*—These are employed at the bottom of the bore-hole, being screwed on to the rods. The simplest and commonest is the flat chisel. It is usually about 18 inches long. It is very easily made and sharpened, and is suitable for all kinds of stone of moderate hardness. The width of its cutting edge is made according to the required diameter of the bore-hole (see fig. 34, 1).

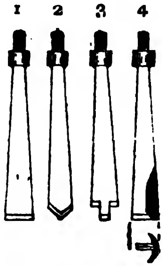


Fig. 34.—Chisels used in Boring

1, Flat chisel. 2, Diamond-pointed or V chisel. 3, Continental V chisel. 4, T chisel.

A chisel suitable for hard rocks is the diamond-pointed or V chisel (fig. 34, 2). It is difficult to sharpen properly. Another very effective chisel upon hard rocks is also termed the V chisel, but it has the disadvantage of being difficult to make and sharpen (see fig. 34, 3). The angle of the edges of the cutting tools is made subject to the hardness of the stone to be bored. A very sharp angle should always be avoided for every kind of stone, because the breaking off of any pieces of the tool in the hole is very mischievous.

To bore through gravel the ordinary chisel is inapplicable, and one with two cutting edges at right angles is used, termed the T chisel. One of the cutting edges is slightly curved (see fig. 34, 4). To bore through soft clay or loose sand an instrument termed an *auger* may be used. It is made a little larger than an ordinary chisel, and being hollow it carries up the core of clay with it (see fig. 35, 1). A wimble (fig. 35, 3) is sometimes used in the same circumstances.

To bore through running sand the ordinary tools are

very unsuitable. A method often adopted is to force a lining of tubes down into the sand, and within these a tube of less diameter with a pointed nozzle at the bottom is put down to the sand. Water under considerable pressure is forced down the inner tube, and stirs up the sand, which rises with the water in the annular space. Both outer and inner tubes are lowered as the sand is removed until the hard stone is reached. The outer tube is then forced tightly upon the stone to prevent any sand getting between the bottom of the tube and stone. The inner tube may then be removed for the boring to proceed in the usual way.

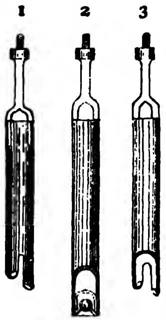


Fig. 35.—1, Auger. 2, Ball-valve Sludger. 3, Wimble.

(2) *Sludgers.*—These are used to clean out the "borings" or debris which accumulates at the bottom of the bore-hole during the process of boring. A sludger is made of wrought iron in a cylindrical form, and fitted with a valve at the bottom, which opens upwards to allow the "borings" to enter, but on being raised it closes and prevents them falling out. It is sometimes termed the "ball-valve sludger", the valve being round (see fig. 35, 2). The common sludger or wimble is slightly different in construction to the ball-valve sludger, as shown in fig. 35. The auger has already been explained.

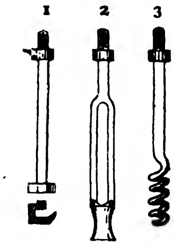


Fig. 36.—1, Crow's-foot. 2, Box-bell. 3, Wadhook.

(3) *Crow's-foot, Bell, and Wadhook.*—In deep bore-holes accidents are liable to occur from fractures of the rods, stripping of screw-threads, pieces breaking off the chisels, part of the rods falling into the bore-hole during the process of taking them out or putting them in, or things falling into the bore-hole. Such accidents

prevent a continuance of the work until the bore-hole is cleared of the broken pieces or rubbish. The *crow's-foot*, shown in fig. 36, 1, is used to extract broken rods from a bore-hole when the fracture is immediately above a joint. The crow's-foot is lowered down to the rods and moved around until it catches hold on the under side of the joint, which is, of course, about double the thickness of the other part of the rods. The whole of the rods may then be lifted to the surface.

When a fracture occurs a few feet above a joint the crow's-foot is inapplicable, because, in raising the rods,

Fig. 37.—Lifting-dog

Fig. 38.—Nipping-fork

Fig. 39.—Screw-key

the long fractured end above the joint would catch the sides of the bore-hole. In such a case the *bell* is employed, and may be either a screw-bell or a box-bell. This is lowered on to the fractured end of the rods, which it grips firmly enough to bring the whole to the surface (see fig. 36, 2). Instead of the bell, the *wadhook* or spiral worm is sometimes employed for the same purpose, and usually acts effectively (see fig. 36, 3).

Tools used in Raising and Lowering Rods.—When rods are to be raised or lowered a *lifting-dog* is used. It consists of an iron clawhook, at the top of which is a ring for attachment to the windlass rope (see fig. 37). When the rods are to be raised the clawhook is placed under the highest joint of the rods after the bracehead is unscrewed. They are then hauled up to the top of the head-gear, and a *nipping-fork* (fig. 38) is placed carefully under the joint which appears above

the top of the bore-hole, to support the rods in the hole when the length already hauled out is being screwed off by means of a *key* (fig. 39). The length screwed off is placed on one side, and the lifting-dog detached and lowered to repeat the operation, until all the rods are withdrawn.

Lining Bore-holes with Tubes.—When clay, sand, or other loose and soft strata are passed through, it becomes necessary to line the bore-hole with tubes in order to preserve it. They are made of wrought iron or steel with screwed joints, the outside diameter to suit the diameter of the hole. The first tube inserted is usually sharpened at the bottom, and is lowered into the hole until its top end is nearly level with the top of the hole; the next tube is screwed on and lowered, and so on until the lining is completed.

Fig. 40.—Wood Plug for withdrawing Tubes

It is obvious that when the boring recommences below a lining of tubes, the diameter of the hole will be reduced by twice the thickness of the iron of the tubes. When a bore-hole requires lining in two or three places the lowest diameter is frequently very small—in some instances too small to proceed with. Sometimes at the finish of a bore-hole it is necessary to withdraw the tubes that may have been put in. This is often a difficult matter, and the cost sometimes exceeds the worth of the tubes. A tool termed a *spring-dart* is used for the purpose, and is dropped sharply into the hole; the spring grips the side of the tube and pulls it out. A wood plug of an ovoid form is sometimes used for the same purpose (see fig. 40). As bore-holes rarely keep perfectly vertical it is often a difficult matter to line them with tubes, and cement grout is now being largely used in place of casing. The grout is pumped down the hollow rods, and after it has set the plug of cement is

bored through and a further section of the hole bored and grouted. Quick setting cements with an addition of 0·33 per cent of soda are those generally used for this purpose.

Special Methods of Boring.

1. *Diamond Method.*—In the methods already described the action of the rods is percussive, that is, by repeated blows, but in this method the action is rotary. The diamond drill consists of a line of hollow rods usually made in five and ten feet lengths, and screwed together. The rods are rotated by an engine through a shaft and bevel gearing, being fed forward at the same time by

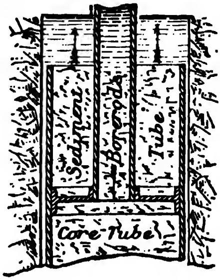


Fig. 41.—Core and Annular Space cut by Diamond Boring Apparatus

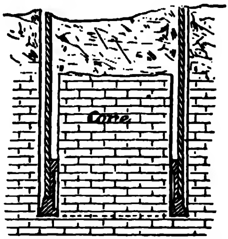


Fig. 42.—Diamond Process of Boring

either a hydraulic cylinder and piston or a screw feed. At the lower end of the rod there is fitted a hollow bit or crown in which pieces of black diamond or carbon are set. When the rods to which this crown is fixed are rotated, the carbons cut an annular space in the rock, leaving the centre-piece or core untouched (figs. 41 and 42). Water is forced down the rods to

keep the carbons cool, and to wash away the cuttings from the bit. If the bit is fitted with water channels, the maximum size of particle that can pass away from the cutter is determined by the size of channel; if no water channels are provided the maximum size of particle that can pass upwards has a diameter equal to the clearance of the outside carbons. A rising current of 12 inches per second is usually sufficient, and to obtain this the quantity of flushing water has to be varied with the size of the hole.

The main feature of this method is the cutting out of a core or section of the rock passed through. At intervals, usually after drilling 10 feet, the rods are withdrawn by means of a hoisting mechanism, and bring with them the core, which is caught and held by means of a self-locking core lifter.

In boring through coal perfect cores are seldom obtained owing to its friable nature, and in order to obtain reliable information as to the thickness of seam, the fine material or sludge washed up by the flush water must be collected. This is done in suitable sludge-boxes, the coal being afterwards separated from the heavier rock material by floating it in a zinc sulphate solution whose specific gravity is greater than that of the coal, and less than that of the rock. Knowing the size of hole, distance bored, size and length of core obtained, and the amount of coal recovered from the sludge, it is easy to calculate the correct thickness of the seam. Diamond drilling is not applied with any marked degree of success in very soft rocks, nor in badly shattered hard rock.

2. *Mather and Platt's Method.*—This method is percussive, but differs from the ordinary method in having a flat hemp or wire rope instead of rigid rods. This rope is wound upon a drum worked by a steam-engine, and the boring tool can therefore be rapidly lowered and raised in the hole. The percussive action is given by means of a small ver-

tical engine placed at the top of the bore-hole. The pulley over which the rope passes from the bore-hole to the drum is attached to the piston-rod, and each upward stroke of the piston raises the pulley, and consequently the rope and cutting tool in the bore-hole—the rope leading to the drum having been clamped. The cutting tool is very ingeniously constructed, and not only bores, but is so contrived as to give to itself a partial turn at each stroke; in fact, it is a self-acting rotary motion. The shell-pump or sludger, for cleaning out the debris which accumulates in the hole, is also of peculiar construction. There is a great saving of time by this method, as the tools can be raised, changed, and lowered very quickly; consequently holes are soon bored. Messrs. Mather and Platt have also introduced a round rope boring apparatus. The head-gear is a timber erection, usually about 60 feet in height. At the top are guide pulleys, and at the bottom of the frame on one side is arranged a winding drum, and on the other a walking-beam which is actuated by a crank driven by a large belt pulley from a steam-engine. The boring-bar is of great length and weight. The walking-beam gives the necessary motion to the boring-bar, which rises and falls according to the stroke. The bar is caused to rotate by the borer twisting the rope on the surface, and at the same time he gradually feeds out the rope as the hole deepens. The boring-tool is withdrawn and lowered by the winding drum.

3. *The Davis Calyx Method.*—This method of boring has been introduced from America, and bore-holes have been successfully put down by it in the search for coal in Kent. The details are somewhat similar to the diamond system, but instead of a diamond crown, a bit or cutter is adopted, consisting of a cylindrical metallic shell, the lower end of which is formed into a series of long sharp teeth, specially arranged to drill the hole just large enough to allow the apparatus to descend freely, and at the same time keep it in a vertical direction. Cores are extracted of the rocks bored through.

Shot Boring.—Steel shot are now frequently used as an abrasive in place of diamonds. The general fit out is similar to that used in diamond boring, but the boring bit, collection of sludge, method of gripping the core, and the feed mechanism differ. The boring bit is formed from a hollow steel cylinder (fig. 43), with a diagonal slot cut in its lower edge. The chilled steel shot are fed through the hollow rods to the bottom of the hole, where they are caught by the bit and driven round under it, thus cutting the strata by a milling action. The core is broken off by dropping angular pieces of quartz down the hollow rods. These jam themselves between the inner side of the rod and the core, and thus detach the core, which is raised with the rods.

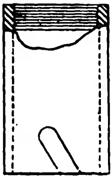


Fig. 43.—Boring Crown for Steel Shot.

The slot in the bit allows an easy passage for the flushing water and prevents any tendency for shot to be washed out of position. If, however, any of the shot work up the side of the hole or the core they are caught by the slot and fall back to the bottom. The chief drawbacks to this system are, first, the difficulty of boring in soft rock due to the shot becoming imbedded and thus stopping the milling action; second, the smallest diameter of bit that can be used economically seems to be about 4 inches; and third, possibility of trouble in very open jointed work.

Combination churn and core drilling is sometimes adopted, the chisel bit being used in soft rock and chilled shot in the harder rocks encountered. This provides more rapid progress, but does not give so much detailed information.

Cost of Boring.—Boring is frequently contracted for, the contractors supplying all necessary appliances and workmen. The following is the usual scale of charges for boring in ordinary coal-measure strata:

For first	5 fathoms,	7*s*. 6*d*.	per fathom.
„ second	5 „	15*s*. 0*d*.	„
„ third	5 „	22*s*. 6*d*.	„

and so on, increasing in price 7*s*. 6*d*. per fathom for every five fathoms increase in depth.

The Diamond Rock Boring Company's price for boring is 8*s*. per foot for the first 100 feet, 16*s*. per foot for the second 100 feet, 24*s*. for the third 100 feet, and so on, increasing by 8*s*. per 100 feet. These prices are subject to fluctuation with changes in cost of material and labour.

In very deep bore-holes, and when stone of unusual and unexpected hardness is encountered, special prices are arranged to suit the circumstances.

The cost of a bore-hole, which is usually let at so much per fathom for the first 5 or 10 fathoms, and increases by a definite fixed sum per fathom at the commencement of each step, may be found from the sum of an arithmetical progression, thus:

Let x = total cost in shillings.
a = price of first step.
d = increment of increase for each additional step.
n = number of steps.

Then $$x = \left\{2a + (n-1)d\right\}\frac{n}{2}.$$

EXAMPLE.—Find the cost of a bore-hole 250 fathoms deep if the price is 10*s*. per fathom for the first 5 fathoms, and increases 10*s*. per fathom for every additional 5 fathoms.

Here $$n = \frac{250}{5} = 50 \text{ steps.}$$

$$\begin{aligned} \therefore x &= \left\{2 \times 10 \times 5 + (50-1)10\right\}\frac{50}{2} \\ &= \{100 + 490\}25, \\ &= 590 \times 25 \\ &= 14750 \text{ shillings or £737, 10s.} \end{aligned}$$

Deep Bore-holes.—The following are examples of deep bore-holes.

The deepest bore-hole which has yet been put down is at Schladebach, near Kötschau, in Merseburg, Prussia, which attained the depth of 956 fathoms. The boring was begun in August, 1880, by the Royal Division of Prussian Mines, and finished in the autumn of 1886, having occupied 1247 actual working days, with a mean daily advance of 4 feet 7 inches, and a total cost of £10,615—or, say, £6 per yard of boring. The initial diameter of the bore-hole was 11 inches, and the diameter at the bottom was only 1¼ inches, it having been reduced by repeated tube linings. Careful observations of temperature were taken as the bore-hole descended; the last reading was taken at a depth of 938 fathoms, and registered 134° F. The bore-hole begins in the Trias, and passes through the Permian into the Devonian.

Another deep bore-hole is at Sperenberg, Berlin. It starts with a diameter of 12 inches and reaches a depth of 695 fathoms. The deepest core drill-hole in America was put down in New Jersey to a depth of 4920 feet.

CHAPTER VI

SINKING OF COAL SHAFTS

When the existence of coal-seams has been satisfactorily proved, and it is decided to sink shafts to work them, it is necessary in the first place to fix upon the position, number, size, and form of the intended shafts.

The Position of Shafts.—This depends upon various circumstances. In every case, however, it is desirable to have the winding or drawing shaft in the vicinity of a railway or navigable river, so that the coals may be readily sent off to the market. A main road is also desirable, but is not of such paramount importance as a railway or river. Before sinking very near to a river, the question of an overflow should be considered, and

also the likelihood of the shaft having to pass through sand and gravel.

Surface and royalty owners usually have to be consulted in the matter, and sometimes the position they decide upon is unsuitable to the underground conditions.

The proper position for the winding shaft should be near the centre of the coal royalty to be worked. Generally speaking, the position that allows about two-thirds of the area to be worked to the rise and the remaining one-third to the dip is the best, but this is influenced by the presence of large faults, dip of strata, and water. When the beds have considerable dip, the shaft should be sunk nearer the lowest point, so that the coals and water will gravitate to it.

Generally, the drawing, pumping, and ventilating shafts are all placed as close together as the Mines Act allows, so as to concentrate the surface works. In some cases the drawing shaft is sunk to be near the surface railway, the pumping shaft to be at the lowest point underground, and the ventilating shaft to be at the highest underground point so that the gases will all ascend to it; but this is impossible where the area of the royalty is large because of difficulties in connecting them for ventilation purposes.

Number of Shafts.—To be in accordance with the provisions of Section 36, Coal Mines Act, 1911, " There must be in every mine at least two shafts or outlets, with which every seam for the time being at work in the mine shall have a communication, so that such shafts or outlets shall afford separate means of ingress or egress available to the persons employed in every such seam, whether the shafts or outlets belong to the same mine or to more than one mine. Such two shafts or outlets must not at any point be nearer to one another than, fifteen yards, and there shall be between them a communication not less than four feet wide and four feet high."

Formerly, in many instances, only one shaft was sunk

and made to serve the purposes of drawing, pumping, and ventilating, but owing to the frequency of accidents whereby men were imprisoned in the mine, notably the beam accident at Hartley Colliery, Northumberland, in 1862, when 204 lives were lost, two shafts were rendered compulsory by Act of Parliament.

Size of Shafts.—This will be determined by the anticipated output per day, the area to be worked, and the depth of the coal-seams. In deep pits producing a large daily output it is very necessary to have shafts of large dimensions, as the winding is accomplished rapidly, and large volumes of air are required to pass per minute to ventilate the mine. It should also be considered that pipes for pumping purposes, electric cables, and boxes for haulage ropes may be put into the shaft at a future date. In the early days of coal mining the shafts were usually of very small size, varying from 5 to 10 feet diameter; now they are rarely less than 10 feet, and often deep shafts are 18 to 20 feet, and sometimes as much as 23 feet diameter.

Form of Shafts.—Shafts may be *circular*, *rectangular*, *elliptical*, or *polygonal.*

The CIRCULAR FORM is generally adopted at coal-mines in England. It is the strongest form, and will resist the pressure of soft clay and running sand better than any other form. It is suitable for the application of cast-iron tubbing to dam back and resist heavy pressures of water, and also for lining with masonry to support weak or crumbling strata. Circular shafts are easy to sink, and although when fitted up with winding arrangements there is a waste of space between the square ends of the cages and the circumference or side of the shaft, yet this space in large mines is essential for the free passage of air for the ventilation of the mines.

The RECTANGULAR FORM is adopted at many of the coal-mines in Scotland. It is also frequently adopted in English mines for staples or small shafts communi-

cating from one seam to another underground where the strata are strong and do not require masonry. In metalliferous mining this form of shaft is very generally adopted, and is usually divided by wood partitions into compartments for winding, pumping, and ladder-way—the latter for the ingress and egress of the miners. In the United States this form of shaft is very largely adopted; timber is very plentiful and cheap, and many of the shafts are lined from top to bottom with oak.

The objections to the rectangular or oblong shaft as compared with the circular shaft are:

(*a*) The labour of sinking is increased, owing to the difficulty of cutting out the four corners.

(*b*) The cost of lining the shaft with timber is considerable.

(*c*) If water is encountered in sinking it is more difficult to shut it off.

(*d*) There is greater difficulty in sinking through soft and faulted ground, and much more cost.

(*e*) Timber lining put into an oblong shaft will not last as long as the brickwork usually put into a round shaft.

The ELLIPTICAL and POLYGONAL FORMS are frequently adopted on the Continent, but offer no advantages compared with the circular form, of which they are modifications.

Sinking through Alluvial Matter to the Stone.—Before commencing to sink it is necessary to procure the various tools and appliances for the special work of sinking. These consist of picks, shovels, hammers, drilling-machines, jumpers, blasting-gear, ropes, spring hooks, &c. Kibbles or hoppits are needed to raise the material and water out of the shafts, for sending bricks, tools, and other things down, and for the sinkers to descend and ascend in them. They are made barrel-shaped, of iron or steel, and provided with a large hoop at the top (see fig. 44), and are attached to the winding-

rope by means of a spring hook. They are of from 30 to 40 feet cubical capacity, and are so arranged as to be easily tipped.

The head-gear arrangements depend upon the depth and diameter of the proposed shafts. In recent years pits have been sunk of large diameters and to great depths, and great attention has been paid to equip the surface erections with substantial head-gears in order to

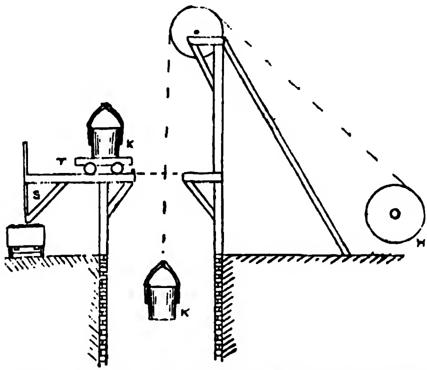


Fig. 44.—Surface arrangements at a Sinking Pit, showing how the material is raised and tipped

W, Winding drum. KK, Kibbles or bowks. T, Banking tram. S, Shoot or hopper for stones raised from pit.

carry on the sinking expeditiously and economically. The general arrangements consist of a square framed wood head-gear about 30 to 40 feet in height, with a landing-stage about 15 feet above surface level to facilitate the discharging of the material sent out of the shaft. The shaft covering consists of a bogie or trolley running on rails to receive the kibbles, or of balanced folding-doors. Sometimes a permanent winding-engine is erected and used for the sinking, but generally a temporary small winding-engine is used. A detaching hook must now be used in case of an overwind. Black-

smiths' and joiners' shops for repairing purposes are also put up temporarily. Very often provision has to be made for pumping water from the shaft, and additional boilers for steam raising will have to be estimated for.

Assuming it is intended to sink a circular shaft, the first operation will be the marking off on the ground the position of the centre and circumference. The diameter at the start is usually made from 2½ to 4 feet more than the actual diameter intended for the shaft, so as to give sufficient room for any supports to the shaft sides that may be required. If the ground consists of soft earth, loam, or clay, the shaft is sunk about 6 feet with pick and shovel, and the earth thrown to the surface. The sinking is then stopped until the shaft side or circumference is supported with timber, in order to prevent any earth sliding into the shaft. The timber consists of *cribs*, *backing deals*, *punch-props*, and *stringing deals*.

Cribs are segments of oak, cut to suit the circle of the shaft, about 4 feet long and 6 inches square section, with pieces of wood called cleats nailed at each end, and the ends forming joints which are in the radii of the circle of the shaft (see figs. 45, 46), or they may be formed from iron hoops, made in segments and bolted together.

The shaft bottom is levelled, and a sufficient number of segments to complete a circle are sent down and laid upon it. Another circle is formed above this, raised about 3 feet, and supported upon short wood props termed punch-props, set upon the bottom circle. Then another circle is placed in the same way 3 feet above the last on a level with the surface, and a fourth one to project about 3 feet above the surface. Wood deals, termed *backing deals*, 9 feet long and 1 inch thick, are placed behind the cribbing all around the shaft and wedged tight. The whole of the timbering may be supported by *stringing deals*, which are nailed on the inside of the cribbing, and if necessary secured to strong balks laid horizontally across the top of the shaft.

The sinking may now be proceeded with for a short

depth, say from 6 to 9 feet according to the nature of the strata, and then additional cribbing is put in as before.

The vertical height that the cribs are placed apart has been given as 3 feet, this being the usual height in moderately firm ground. The height to a great extent depends upon the nature of the ground and the pressure. In some cases the pressure is so great as to necessitate the cribs being within 6 or 9 inches of each other.

Fig. 45.—Front View of two Segments of Cribs

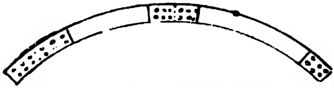


Fig. 46.—Plan of two Segments of Cribs, showing the wood cleats at the joints

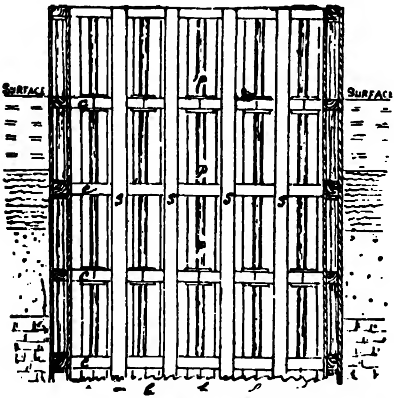


Fig. 47.—Section showing Shaft Timbering

c, Oak cribs. *p*, Punch-props. *s*, Stringing deals. *b*, Backing deals.

The depth, 12 feet, is now too much to allow of the earth being thrown to the surface, and the kibble will now be utilized to raise it up. The sinking and timbering gradually proceed in the way described until the "stone-head" or the first rock-bed is reached. If this bed is of a soft nature, the sinking is continued through it until a hard bed is reached; then in order to make the shaft secure it is necessary to build a circular wall and remove all the timbering.

Walling the Shaft.—The walling is commenced at the bottom of the shaft on a level bed prepared on a

hard stone. A walling or wedging crib of wood or iron is first placed on the level bed, and the walling rests upon it. This crib is made up of segments about 4 feet long, 14 inches wide, and 6 or 8 inches deep, the whole carefully laid and tightly wedged against the shaft side, with its centre coinciding exactly with the centre of the shaft. Sometimes the shaft is widened and a ring consisting of at least six courses of brickwork is built hard against the side, any spaces being completely built up. This ring should be not less than 2 feet 3 inches thick, and when so constructed it serves as a foundation for the remainder of that section of walling. Walling is sometimes done with fire-bricks, but more often with fire-clay lumps, and occasionally with prepared stones. The fire-clay lumps make a better wall, but are more expensive than ordinary fire-bricks (see fig. 48).

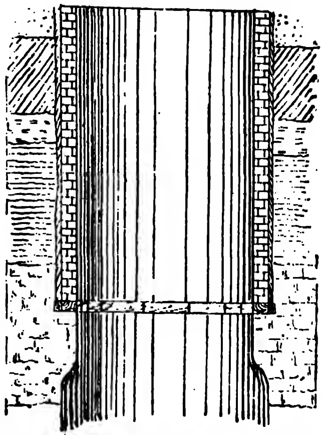


Fig. 48.—Section showing Shaft Walling

As the walling rises from the shaft bottom the masons have to stand upon a wooden platform called a *cradle* (see fig. 49), suspended by a rope from the surface, by which it can be raised and lowered. The cradle is made circular, of about 6 inches less in diameter than the shaft, so that it may work freely up and down and allow the space below to be ventilated. It is made in two portions strongly bolted together, as it has to support the weight of a number of men and a quantity of walling material. It is hung from the rope by six chains placed an equal distance apart, or made to rest by strong iron

sliding bolts upon the wall sides, and raised every 5 feet as the walling is built up.

When the shaft sides are wet, water-rings are built in the walling at intervals for the purpose of preventing the water dropping down the shaft. (See fig. 50.) Sometimes the water-ring is similar to the walling crib, but with a groove or channel cut in it right round the shaft, and sometimes it consists of fire-clay lumps with a groove moulded in them. The brickwork above the ring is cut back a little, so that the drops coming down the shaft

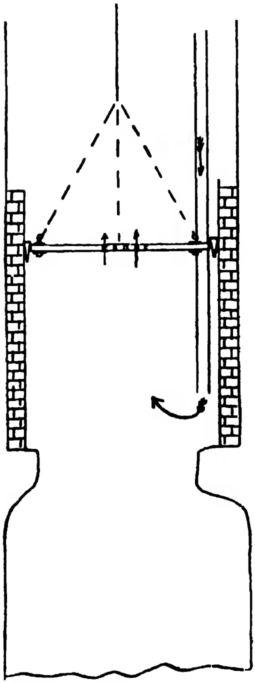


Fig. 49.—Platform or Cradle upon which men stand to build Shaft Walling. Also Air-pipe to ventilate underneath Cradle

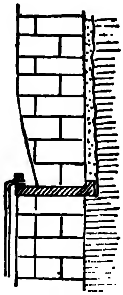


Fig. 50.—Cast-iron Water-ring or Garland built in Shaft Walling, and Pipe to conduct Water down Shaft

sides will run into the water-ring and then through a pipe to the shaft bottom. While the walling is being built, as much as possible of the timber is removed, and any space there may be between the wall and the earth is packed with ashes. When the walling is com-

pleted the sinking may be resumed in the stone. (See page 66.)

Sinking through Sand by Piling.—This is an old way of sinking through a quicksand, and was adopted at Framwellgate Moor Colliery, near Durham, in 1838, to get through 144 feet of sand. The piles consist of wood planks 10 to 15 feet long and 6 inches by 3 inches section, and are sharpened at the driving end and shod with iron (see fig. 51). They are driven vertically down side by side into the sand around the circumference of the shaft marked off on the surface. Some of the sand is then dug out, and a crib put on the inside of the piles to support them; then a little more sand is dug out and another crib put in, until about 3 or 4 feet off the bottom of the piles it becomes necessary to drive down another tier. This tier is driven down within the cribbing, and the process continues until the stone-head is reached. Each fresh tier of piles reduces the diameter of the shaft by twice the thickness of the cribbing and piles; therefore it is necessary to ascertain the thickness of the sand, and calculate the number of tiers, before commencing to sink, so that the initial diameter of the shaft may be large enough to give the proper diameter when the stone-head is reached.

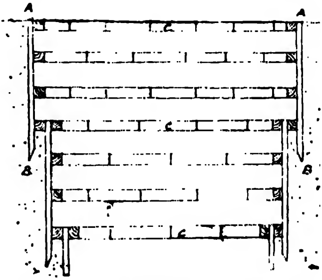


Fig. 51.—Pile Sinking through Sand and Gravel

A B, Wood piles. c, Cribs.

In the Continental method the piles used are only about 3 feet long and 4 inches by 1 inch section. Instead of being driven vertically down they are given a slight outward inclination. The sand is excavated and cribs put in until a depth of 2 feet 6 inches is reached, then

another course of piles is driven down, and so on. The chief advantage of this method is that the shaft does not require to be much increased in diameter beyond the normal, and consequently there is less sand to remove and less pressure to resist. A recent example (1906) of shaft sinking by piling is that of Bowburn Winning, near Durham, which went through 156 feet 9 inches of Glacial Drift, consisting of clay and sand, before any solid rock was reached.

Sinking through Sand with Wood or Cast-iron Cylinders.

With Wood Cylinder.—This is made of oak cribs placed at short intervals apart with thick backing deals bolted to the cribs, and forming as nearly as possible a water-tight casing. The cylinder is provided with a cutter or leader of iron 3 feet deep, sharpened all around its lower edge to enable it to sink into the sand. The length of the cylinder depends on the thickness of the sand-bed. When the cylinder is not of sufficient weight in itself to cause it to sink, it is usually weighted with bricks or pig-iron. The sand is carefully extracted from the inside of the cylinder, and as this is done the cylinder sinks until the stone-head is reached and all the sand removed. When a hard stone is reached, upon which to lay a walling crib, the shaft is permanently walled up through the sand-bed.

With Cast-iron Cylinder.—This is made of cast-iron segments with the flanges and ribs on the inside; with ordinary tubbing they are on the outside. The segments are bolted together, and attached to the bottom of the cylinder is a cutter sharpened at the bottom end to make it sink easily. Sometimes the cylinder is of sufficient weight in itself to sink as the sand is being extracted, but often it requires weighting; this is done by constructing inside of it a scaffold, which is loaded with bricks or pig-iron. As the cylinder descends segments are added to it until the sand is got through and the stone-head reached. When a hard bed is found, a

wedging crib is prepared, the cutter removed, and the tubbing made complete.

Generally the cylinder is attached at the top to chains and screws from four balks at the surface, and it is lowered as required by the screws. The great objection to the use of cylinders is the difficulty of keeping them in the perpendicular line as they are being forced down, and it is very important that the shaft should be plumb. When the cylinder leaves the perpendicular it may be drawn back by shifting the pig-iron ballast over to the side that is dragging behind. Where this has to be done the scaffold carrying the ballast must be close down to the bottom end of the cylinder.

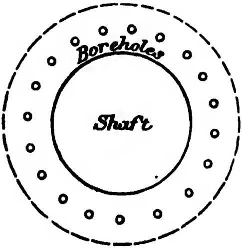


Fig. 52.—Arrangement of Bore-holes for Freezing Process

Herr Poetsch's Freezing Method.—This method of sinking through quicksand and other soft ground containing much water has been tried with much success, the principle being to freeze the ground for some distance around the position of the shaft into a hard, solid mass, which can be sunk through in the ordinary way. The process is to put down several bore-holes in and around the shaft to the stone-head (see fig. 52), and in each bore-hole to put a wrought-iron tube closed at the bottom end, and in each tube to place a smaller tube with two slotted openings opposite each other at the bottom end. The joints of these tubes, especially the outer tubes, must be perfectly water-tight. The tops of the small tubes are connected to the delivery end of a force-pump, which forces the cooling fluid, a solution of chloride of calcium, cooled considerably below the freezing-point of water, down them through the openings at the bottom into the larger tubes, up which it ascends to the refrigerator to be again reduced in temperature

and pumped into the pipes once more. The freezing mixture is reduced in temperature below zero by means of ammonia. The effect of the cold fluid passing up and down the bore-hole tubes is to freeze the sand and water, and form solid ice cylinders around each bore-hole until they become joined, and so constitute one solid, firm mass of frozen ground. This may then be sunk through to the hard stone, from which walling or tubbing is built up to the surface. The freezing process must be kept up around the shaft until this is completed.

Gebhardt and Koenig's Improved Freezing Process.—This process consists of a number of bore-holes at a given distance from the centre of the contemplated shaft, which are bored through the wet beds or quicksands into the solid rock. These bore-holes must be absolutely vertical and serve to receive the freezing tubes, which latter are hermetically sealed and contain inner tubes, open at the bottom. Refrigerating machines send the freezing-medium through the inner tubes, whence it enters from below into the outer sealed tubes, and, whilst slowly ascending in these, communicates a portion of its cold to the surrounding ground.

When reaching the top it re-enters the refrigerating machines, is cooled, and sent on the same course over and over again, each time imparting further cold to the surrounding ground. This process is continued uninterruptedly for a longer or shorter period—depending entirely upon the formation of the strata and the dimensions of the shaft—until a strong frost wall has formed and completely closed around the contemplated shaft. This frost wall is maintained until, under its protection, the sinking of the shaft and the walling with iron tubbings or brickwork have been completed fully and in thorough safety. This method has been successfully applied in this country and in Germany to sink through quicksands and water-bearing strata where the ordinary methods have been inadequate.

Goebert's System.—Owing to trouble with leakage, especially where the strata have to be frozen to some depth, and difficulties of ascertaining whether the circulation of the cooling liquid is satisfactory, the foregoing methods have been largely superseded by that devised by Goebert. Here the arrangement of bore-holes and inner tubes is similar to that used by Poetsch, but in place of a cold solution of calcium chloride ($CaCl_2$) being used cooling is effected by pumping liquefied ammonia (NH_3) down the inner tubes. These are usually made of copper in the form of a spiral with fine perforations through which the liquid issues as a spray. The liquid evaporates and this evaporation takes up heat from the surrounding strata, and in doing so cools it below the freezing-point of water. The ammonia gas produced by evaporation is drawn off by a pump, reliquefied by compression, and used over again. As the pressure inside the tubes lining the bore-holes is less than that outside any water coming in is at once frozen and the leak stopped. The greatest amount of evaporation is at the top until freezing is set up, hence the strata are frozen from the top downwards which admits of sinking operations being begun almost at once, whereas with the other systems described freezing takes place from the bottom upwards, so that the whole depth must be frozen before sinking can be commenced. In this as in all other freezing systems, once the water-bearing beds have been penetrated by the sinkers the shaft must be lined with water-tight tubbing to hold back the water, which for the time being has been held back by a wall of ice.

Method by Injection of Liquid Cement.—Another method for consolidating quicksand in water-bearing strata consists in injecting fine liquid cement, by means of compressed air, into the sand in the area of ground to be consolidated. The cement, which must be very clean and fine, is taken by an injector, which forces it through a flexible pipe into a perforated tube

sunk in the sand to the required depth. Two methods are in use, the long hole method where bore-holes are put down from the surface, and the short hole method where the cementation is effected through short holes carried some 9 or 10 feet ahead of the sinking operations.

Timbering Rectangular Shafts.—There are several ways of arranging the timbers for supporting the sides of rectangular shafts, but the simplest is that of sets composed of four corner-posts and four intermediate posts from four to six feet in height, supporting

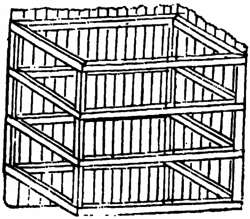


Fig. 53.—Timbering of a Rectangular Shaft

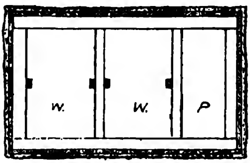


Fig. 54.—Plan showing Timbering of a Rectangular Shaft

W W, Winding compartments. P, Pumping compartment.

a horizontal set of timbers of square section (see figs. 53, 54). The sets are placed one above another, and at intervals of 30 feet or so very strong pieces of timber are notched into the shaft sides to support the weight of the sets above. Behind the sets of timbers, the shaft is lined along each side and end with planks about 2 inches thick placed vertically, and the space, if any, between the planks and the stone is filled with soil or ashes. The ends of each of the four pieces of timber used in a horizontal set are cut to a pattern, so that they fit easily and closely together when placed in the shaft. The sectional dimensions of the timber used depends upon the length and width of the shaft, and upon the character of the strata to be supported. Rectangular shafts are usually divided by means of buntons, to which planks are secured, into two or more compartments,

according to the requirements of the mine. Frequently three compartments are made—two for winding and one for pumping, as in fig. 54, but occasionally a fourth compartment is required for a ladder-way. It has been found that a rectangular shaft is stronger when made long and narrow, and the compartments arranged side by side, as shown in fig. 55, than when there is very little difference between the length and the width and the compartments arranged as in fig. 56. The verticality of the sides and of the timbering of a rectangular shaft is secured by suspending a plumb-line at each corner.

Fig. 55

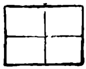


Fig. 56

REGULATIONS FOR ENSURING SAFETY IN SINKING

For the purpose of the regulations, kibble includes kettle, hoppit, tub, bowk, barrel, or cage.

In addition to the daily examination required by Section 66 of the Act, the master sinker, or a competent person appointed for the purpose by the manager, shall once at least in every twenty-four hours, examine thoroughly the state of the shaft and the state of all gear by which cradles, platforms, or pumps are slung in the shaft, or by which persons or material are raised or lowered.

No person shall ride on or against a full kibble or on the edge of a kibble.

Every cradle or platform used in the shaft shall be constructed with a grid or other suitable contrivance, when necessary to secure the efficient ventilation of the whole of the shaft.

Every cradle or platform on which men work in the shaft shall be so protected as to prevent anyone falling off.

While men are at work on any cradle or platform in the shaft the following precautions shall be strictly observed:

The cradle or platform shall be secured to the sides of the shaft in order to prevent its swinging.

The flap over the kibble hole shall be securely fastened.

If the cradle or platform is constructed of two or more pieces hinged, the pieces shall be securely bolted together.

The cradle or platform shall not be moved except by the express direction of the manager, master sinker, or chargeman.

The examination required to be made by the chargeman before the commencement of work shall be made immediately before the descent of the shift.

The chargeman shall as part of his examination before the commencement of work, or if work is carried on without any interval by a succession of shifts, then as part of his examinations during his shift, examine carefully the sides of the shaft, take off any loose stones, and otherwise satisfy himself that the shaft is in a safe condition for men to work at the bottom. When men are engaged in walling or tubbing the shaft a similar examination shall be made by a competent person appointed by the manager.

The chargeman shall be the last man to ride at the end of his shift, and, if his shift is succeeded immediately by another shift, he shall not leave the bottom of the shaft until after the descent of the chargeman of the next shift.

No person shall be allowed to descend after any cessation of work in the shaft caused by the withdrawal of the workmen for shot-firing or other purposes until the chargeman, accompanied if necessary by not more than two other persons, has descended and examined the shaft, and found it to be safe in all respects. If inflammable gas has been found, or is likely to be found in the shaft, the examination shall be made with a locked safety lamp of a type which will indicate the presence of such gas.

When lowering the kibble the winding engineman shall stop it when it has reached a point three fathoms

above the bottom of the shaft or above any cradle or platform upon which the kibble is to alight, and shall wait the signal from the chargeman to let it down. When raising the kibble he shall stop the engine as soon as the kibble has been raised four feet from the bottom, in order that the chargeman may see that the rope is steadied, and shall not again move his engine until he has received the signal from the banksman or chargeman.

When gear, tools, or materials are being lowered the banksman shall see (*a*) that the kibble is properly loaded; (*b*) that no loose material is packed above the level of the top of the kibble; and (*c*) that gear or tools are put into an empty kibble, and if they project above the level of the top are securely fastened to the bow or chains of the kibble; and (*d*) that timber and other bulky articles are safely hung.

CHAPTER VII

SINKING THROUGH STONE; TUBBING OF SHAFTS; AND KIND-CHAUDRON METHOD

Process of Sinking after the Stonehead is Reached.—When the stonehead is reached and the shaft above made secure and safe by walling or tubbing, the sinking may be resumed. For the first few feet below the walling crib the shaft is sunk down in a line with the inside of the walling, that is, it is made of the same diameter as the walled part, which is the finished size. It is then sheared back, or gradually enlarged in diameter to the full size, to admit of any future walling or tubbing which may be needed, and continued at this increased size. This is done in such a way that the brickwork is not undermined, but rests upon a stone shelf or bracket sufficiently strong to support it during

the further sinking of the shaft, until a depth has been reached at which it is necessary to put in additional walling.

When any more walling is put in, the stone which was left as a foundation for the walling above is gradually removed, and the lower course joined up to the walling crib of the upper course, so as to form a complete shaft wall.

The rocks of the coal measures are usually too hard to be sunk through without blasting operations. Holes are bored or drilled downwards in the stone at the shaft bottom, the first holes being placed near the centre, and heavily charged with an explosive to blow out a *sump*. The number, position, and depth of these holes, and the outer circle of holes, naturally varies according to the diameter of the shaft and the nature of the rock being passed through.

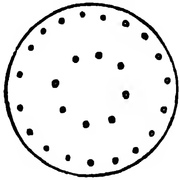


Fig. 57

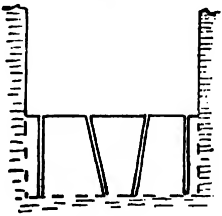


Fig. 58

Figs. 57, 58.—Plan and Section of Shot-holes in bottom of Sinking Pit

In very hard sandstone, as the pennant rock of South Wales, nine *sumping-holes* are usually allowed about 4 to 5 feet deep, and from twenty to twenty-four outer circle or *canch-holes* round the sides about 4 feet deep for a shaft of 15 to 18 feet diameter.

In ordinary sandstone the holes would be drilled about one foot deeper. The number of sumping-holes would be the same, but the canch-holes required would be reduced one-third or more.

The drilling of the holes is sometimes done by one man sitting down and holding the drill, while other two do the striking necessary to drive it downwards, the holder turning it round a little after each blow. More

often, however, the holes are put in by drilling machines worked by (a) manual labour, (b) compressed air, or (c)

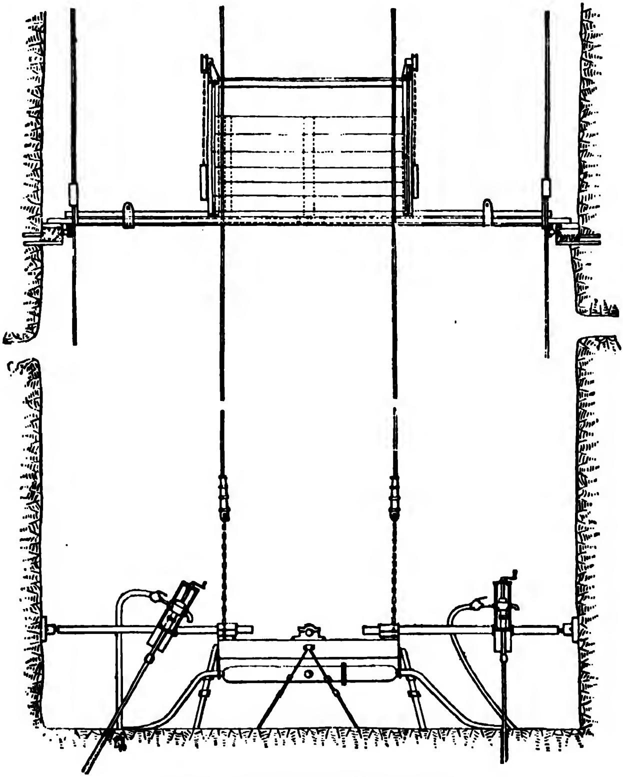


Fig. 59.—Walker's Shaft-sinking Frame

electricity. The machines may be worked on ordinary tripod stands, or from a heavy circular iron frame with screws to set out against the shaft side to keep it steady. Walker's shaft-sinking frame (see fig. 59) is now used to

carry the compressed-air percussive drilling machines. There are many forms of steel bits used for rock drills. For hardstone + (fig. 60*a*) or × (fig. 60*b*) bits are most suitable. For sandstone, bits with a broad edge (fig. 60*c*); and for soft ground the form shown in fig. 60*d* is most suitable, as it keeps the hole circular without much difficulty.

The operation of charging and firing a shot is described in Chapter X on Explosives.

Dynamite or gelignite are the explosives most gener-

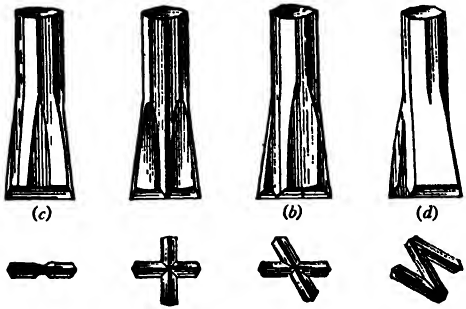


Fig. 60.—Bits for Drills

ally used in blasting in sinking shafts; with gunpowder there is considerable difficulty and loss of time, especially in wet shafts, in " getting the shots off ". Great care should be taken when lighting up shots. All tools should be sent to the surface, and only two chargemen left to light up the shots. They should see that everything is ready, and the kibble standing on the bottom for them to get into and ascend immediately the shots are lighted. The reports of the shots are to be carefully counted to ascertain if all explode, and if any are missed it is unsafe for anyone to descend for a considerable time after the last shot heard.

The application of electricity for lighting up shots is a great improvement upon the method described above.

The cables are carried down the shaft from the surface and attached to the detonators in the shot-holes. A number of shots can be fired together; the effect from a given amount of explosive is greater, and more stone is simultaneously removed than by the old method. The advantages claimed for the electric system of shot-firing in shaft-sinking are:—

(*a*) All men are brought to the surface before the wires are connected to the exploder; therefore it is safer.

(*b*) If all the shots do not explode, there is no risk to anyone descending as soon as the wires are disconnected from the exploder.

(*c*) The work can be done with greater dispatch and cheapness.

(*d*) Absence of smoke or fumes soon after the shots are fired.

Where both sumping and side holes are drilled, the sumping holes are charged and fired first. An iron rod is put into the side holes to prevent them being choked up. While the stones from the sumping holes are being cleared away, the side holes are being made ready to be fired as soon as convenient. In any shaft which is not intended to be walled, no shots ought to be placed nearer the shaft side than one foot; this foot of stone is generally taken off all around the shaft with picks and wedges to dress and smooth the side.

Cast-iron Tubbing to dam back Water in Shafts.—It frequently happens in sinking shafts that large feeders or quantities of water are given off by the beds, and it is necessary to have them dammed back in order to prosecute the sinking, and to avoid future pumping as much as possible. Several methods have been tried to effect this, but the most satisfactory is to line the shaft with cast-iron or metal tubbing. (Fig. 61.)

The tubbing is made in segments, cast to suit the *sweep* or circumference of the shaft, generally about 4 feet long, from 18 to 36 inches in depth, and of a

thickness varying according to the depth of the surface. One side of the tubbing is quite smooth, the other side, which faces the stone, is strengthened with ribs, flanges, and brackets. Each segment has a flange projecting on the top and down one side to keep the adjoining segments in position. In the middle of each segment is a small hole to allow of the escape of water and gas during the fixing of the tubbing; this hole is also utilized for attaching a bolt and chain when lowering and placing each segment in its position in the shaft.

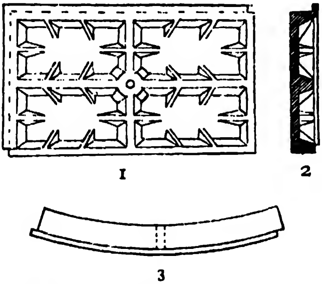


Fig. 61.—1, Back view of a segment of Cast-iron Tubbing, showing ribs, brackets, flanges, and centre hole. 2, Cross-section of segment. 3, Plan of segment.

When tubbing is to be put into a shaft, the first thing to be done is to select a suitable stone upon which to lay the wedging or tubbing crib. This bed is carefully levelled and tested with a straight-edge and spirit-level. The wedging crib is sometimes made of segments of oak, but more often of cast iron. When of cast iron, it is cast hollow, the open side usually facing the shaft, and supported at intervals by ribs. The size of wedging crib usually adopted is 12 inches in width, 6 inches in depth, and about 4 feet in length. A thin sheeting of wood is first laid on the bed, then the segments are fitted together end to end and a sheeting of wood is placed between the joints. The crib is then made secure by driving wedges between it and the shaft side, and also wedging the joints. Great care should be taken in doing this, as it is very important that the centre of the crib, when ready for the tubbing, should coincide exactly with the centre of the shaft.

When the wedging crib is finished the segments of

tubbing are sent down and laid upon it until a circle is completed, when the second circle is formed by laying segments on the top of the first, and so on until the water-bearing strata is passed through; then a crib is laid on the last circle and wedged tight up against the stone above. The whole of the tubbing is then made tight and secure by filling every vertical and horizontal joint with wedges, commencing with the bottom course. When the tubbing is completed, the plugging up of the holes in the middle of the segments is proceeded with by commencing at the lowest course. As each plug is driven in, it is cleaved with a chisel and a wedge driven in to make it tight. A vent pipe is usually fixed in the upper course of tubbing, so that any air or gas behind the tubbing may be allowed to escape, otherwise the pressure may become so great as to blow out some of the tubbing. The space behind tubbing is usually filled with concrete.

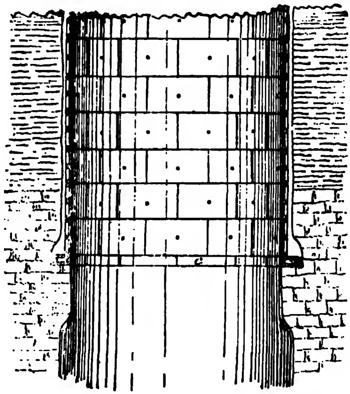


Fig. 62.—Shaft tubbed with Cast-iron Tubbing

c, Wedging crib of cast iron upon which the tubbing is built.

Cast-iron tubbing is very expensive; its cost per fathom depends upon the diameter of the shaft and the depth from the surface. The larger the diameter the more segments are required, and the greater the depth the greater will be the necessary thickness of metal. It is also very difficult to cast the metal of a perfect and uniform strength and quality throughout, consequently in calculating the required thickness of metal, ample

allowance must be made in the formulæ for honeycombs and other imperfections.

Since 1 cubic foot of water weighs 62·5 lb. the weight of a cubic inch of water is $\frac{62 \cdot 5}{1728}$ lb. $=$ ·036 lb., and a column of water 1 foot high and standing on a base of 1 sq. inch weighs ·036 × 12 or ·432 lb. The pressure of water at any depth is therefore: depth in feet × ·432; hence the pressure upon shaft tubbing may be calculated when the head of water is known.

To ascertain the thickness of shaft tubbing when made from cast iron we may use the following formula due to Greenwell:

x = Required thickness in *feet*.
P = Pressure due to vertical depth in feet.
D = Diameter of shaft in feet.

$$x = \cdot 03 + \frac{PD}{50{,}000}.$$

EXAMPLE.—Find the thickness of cast-iron tubbing required to withstand a head of 400 feet of water in a shaft 20 feet diameter.

Here P = 400 × ·432 = 172·8, say 173 lb. per sq. inch.

$$\begin{aligned} \text{and } x &= \cdot 03 + \frac{173 \times 20}{50{,}000} \\ &= \cdot 03 + \cdot 0692 \\ &= \cdot 0992 \text{ feet} \\ &= \cdot 0992 \times 12 = 1 \cdot 19 \text{ inches.} \end{aligned}$$

The following formula for calculating the thickness of tubbing, walling, or of a cylindrical dam to resist pressure of water is given by W. Steadman Aldis in a paper on "Internal Stress in Cylindrical and Spherical Dams":

Cylindrical Dam, Walling, or Tubbing—

K = Thickness in inches.
r = External radius in inches.
T = Ultimate crushing strength in pounds per square inch (see Table).
P = Head of water in pounds per square inch,

$$K = r\left\{1 - \sqrt{1 - \frac{20P}{T}}\right\}.$$

10 is taken as the factor of safety, and is allowed for in the formula. Note that the factor $\frac{20P}{T}$ must be less than 1, otherwise the material proposed has not sufficient compressive strength to be employed for the purpose.

TABLE (Molesworth)

Wrought Iron		T =	38,080
Cast Iron		T =	107,520
Beech		T =	8,500
Oak		T =	10,000
Pitch Pine		T =	6,500
Brick (ordinary red) ..		T =	800
,, (Stourbridge fire) ..		T =	1,717
Sandstone		T =	2,185 to 7,884
Concrete	about	T =	2,000

Every segment of tubbing before being placed in position in the shaft should be carefully examined and tested as to its soundness.

EXAMPLE.—A dam formed from logs of pitch pine has to be placed in a roadway to hold back water at a pressure of 260 lb. per square inch. If the dam is cylindrical and 12 feet radius, find the thickness.

Here $r = 12 \times 12 = 144$ inches, $P = 260$ lb., $T = 6500$, and substituting these values in the formula,

$$
\begin{aligned}
K &= 144\left(1 - \sqrt{1 - \frac{20 \times 260}{6500}}\right) \\
&= 144\left(1 - \sqrt{\frac{6500}{6500} - \frac{5200}{6500}}\right) \\
&= 144(1 - \sqrt{0{\cdot}2}) \\
&= 144(1 - 0{\cdot}45) \\
&= 144 \times 0{\cdot}55 \\
&= 79{\cdot}2 \text{ inches, or } 6{\cdot}6 \text{ feet.}
\end{aligned}
$$

Corrosion of Tubbing.—Tubbing is subject to corrosion by (*a*) acid water, and (*b*) furnace fumes.

(*a*) The effects of acid water are increased by pressure, and tubbing is always subject to more or less pressure. A thick coating of tar or paint may protect it to some extent, but this can only be renewed on the inside.

(*b*) Formerly a furnace to produce ventilation was in some cases built at the bottom of a shaft. The fumes produced by the combustion of the coal, owing to their sulphurous nature, sometimes set up very rapid corrosion in the tubbing. To prevent this a lining of brick was built in front of the tubbing; this plan, however, also had its drawbacks, as it prevented the tubbing from being examined, and leakages could not be got at to wedge.

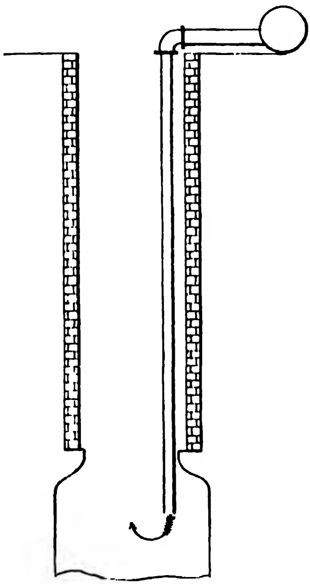


Fig. 63.—Air-pipes to ventilate Sinking Pit. Fresh air forced down the pipes and returns up shaft

Ventilation of a Sinking Pit.—When the shaft has reached a depth of from 60 to 80 feet it will become necessary to make arrangements for the circulation of fresh air to the bottom where the men are working, owing to the vitiation of the air by the fumes of the explosives used, by the gas that may exude from the rocks, and by the breathing of the men. Sometimes wood boxes about 12 inches square are fixed down the side of the shaft. At the top a wind-cowl is placed with its open side facing the wind. The pressure forces the air down the boxes to the bottom, and it returns up the shaft. A better plan is to connect the boxes to a chimney, up which smoke and heat from boilers is passing. This causes an upward draught, and the fresh air passes down the shaft and returns up the boxes to the chimney. Another method is to connect the boxes to a small fan

driven by hand. This causes a current to descend in the shaft and return up the boxes.

These methods, however, are only suitable for small shafts of no great depth. In shafts of large diameter, which are expected to go very deep, it is necessary that a current of air—sufficient to quickly clear away noxious fumes and gases, that are almost sure to be encountered in greater or less quantities—be kept circulating. In some cases large sheet-iron pipes are carried down the side of the shaft and connected to a small fan driven by steam power. In this way a good current of air may be produced, passing down the shaft and up the pipes to the fan. Instead of a fan, the pipes may be connected to an air-compressor, which forces cold air down the pipes to return up the shaft (see fig. 63). The air-compressor may serve the double purpose of ventilating the shaft and driving the drilling machines used in the drilling of holes for blasting purposes.

In some cases the shaft is bratticed or partitioned into two unequal portions, the larger portion acting as the *downcast*, or for the descending air, the other being the *upcast*, or for the ascending air. The brattice may be constructed by fixing two lines of stringing planks, 7 inches wide by 2½ inches thick, with spikes to opposite sides of the shaft. At intervals of 3 feet, wood cross pieces termed *buntons* are notched at each end into the stringing planks, then grooved deals are nailed close together to the buntons all the way down the shaft, so as to make as nearly as possible an air-tight partition.

The Kind-Chaudron Method of Sinking.—This process is really one of *boring*, rather than sinking, the sole object being to put down shafts through hard rocks, where water is lodged in such large quantities that ordinary pumping arrangements are quite inadequate to deal with it. The principle of the method is to effect the sinking, or rather boring of the shaft, without pumping any of the water, and afterwards to dam off the water by means of tubbing, after which the water standing in the shaft

is pumped out, and the sinking resumed in the ordinary way. No men are required in the shaft; they stand upon a wooden platform on the surface to give the cutter a slight turn at each stroke. The shaft is bored in two operations; first of small diameter, and then of the full diameter. It is done by a cutter known by the name *trepan* (see fig. 64). The trepan is suspended by wooden rods from one end of a heavy wood lever, at the other end of which is an engine, which works the trepan. After the trepan has worked for some time it is hauled up, and a sludger or *spoon* is lowered to bring up the debris produced by the boring.

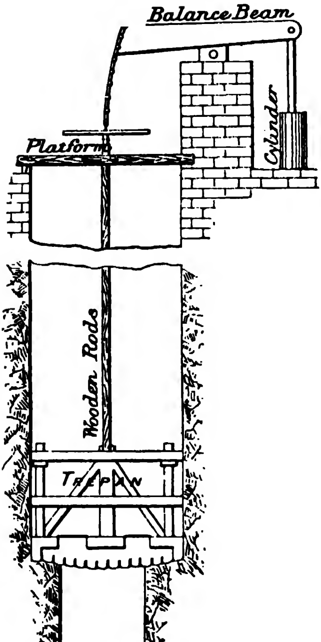


Fig. 64.—Kind-Chaudron Method, showing large Trepan

Another important feature of the method is the tubbing, which is lowered into the shaft to shut off the water. To control the great weight of the tubbing, which is built in complete rings like a huge line of pipes, a diaphragm is placed at the bottom. This floats the tubbing, and it is gradually lowered by allowing water to enter above the diaphragm through a small inner pipe fitted with control valves. The water-tight joint is made by means of a telescope piece at the bottom, the space between the outer flanges of which is packed with moss or

oakum. When the tubbing is lowered on to its seat the heavy top part settles down and compresses the packing between the flanges and against the side of the shaft, thus forming a water-tight joint. The space between the outside of the tubbing and the side of the shaft is then filled with cement.

The method has been tried successfully in England, at Marsden, in the county of Durham. Sinking was commenced in the ordinary way, but very soon such enormous quantities of water were met with that the Kind-Chaudron method was adopted. About 72 yards of water-bearing strata were bored through and tubbed, after which sinking on the ordinary plan was resumed.

For information on the methods of dealing with water given off in sinking shafts see Chapter XXIX.

CHAPTER VIII

SHAFT BOTTOM ARRANGEMENTS; SHAFT PILLARS; DRIVING AND TIMBERING OF LEVELS AND DRIFTS, ETC.

Water Standage.—When the lowest down coal seam which it is intended to work has been reached, it is customary to continue the sinking of the shaft a few fathoms farther, in order to provide a place for the water of the mine to accumulate, and from which the pumps can lift it. This continuation is termed the *sump*, and affords *standage* for the water. Sometimes drifts are driven in the stone from the bottom of the sump, or special places arranged in the seam to provide standage to serve for a few days in case of a breakdown of the pumping machinery.

Shaft Pillars.—The first thing necessary on the opening out of a coal-seam is to make a communication

between the two ventilating shafts, in order to comply with the Coal Mines Act, 1911, which stipulates, " there shall be between them a communication not less than four feet wide and four feet high ".

This communication, which connects the downcast and upcast shafts for ventilative and other purposes, need not be a straight line, but the most suitable course suggested by the circumstances may be taken.

Before commencing to work out the seam or seams of coal, it is necessary to determine the area of coal which must be left unworked, except for the necessary roadways for transport and ventilation, around the shaft bottom in order to preserve the shaft from any disturbance and maintain the stability of the strata in its vicinity. It is obvious that if the coal is removed from around the shaft bottom, he upper strata will subside and probably damage, if not destroy, the shaft and also surface erections around the mouth of the shaft. Accordingly, it is customary and necessary to leave a sufficient area of coal around the bottom of the shaft, and also in every seam of coal to be worked from the shaft, to protect it and prevent any movement of the strata, crush of the coal, or other disturbance likely to cause damage either to the shaft itself or to the engine-houses and other surface erections near the shaft. The area of unworked coal left for this purpose is termed a *shaft pillar* (see fig. 65). The dimensions of such pillars vary according to the following conditions:

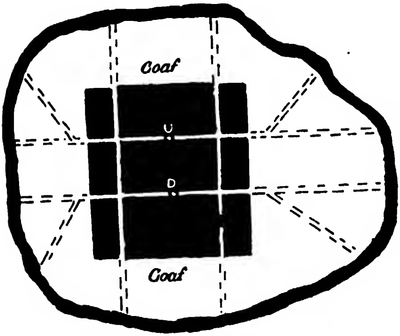


Fig. 65.—Plan showing Shaft Pillar

1. The depth of the coal from the surface. A deep shaft requires a larger pillar than a shallow one.

2. The nature of the coal, roof, and floor. A seam with a soft coal, and a hard top and yielding floor will require a larger pillar than will a hard coal at similar depths.

3. The thickness of the seam.

4. The angle of dip.

5. The existence of any faults in the vicinity of the shaft.

It will be readily understood that no hard-and-fast rule can be arrived at for determining the exact size for shaft pillars to suit every case; circumstances vary so much that no two mines are exactly alike, and the special conditions and surroundings of each case must be studied, and comparisons made with mines of similar conditions, in order to arrive at the most suitable and reliable dimensions.

One approximate rule is: *the depth of the shaft in yards multiplied by* 0·6, *equals the diameter of the area of coal to be left unwrought.*

EXAMPLE.—What size of shaft pillar ought to be left in a seam 300 yards deep?

$$300 \times \cdot 6 = 180 \text{ yards diameter.}$$

Taking the shaft as centre and radius 90 yards, the circle described is the size of the shaft pillar at a depth of 300 yards.

The foregoing rule applies particularly to cases where the shaft penetrates ordinary coal measures strata. If, however, there should be more than a few feet of surface, an additional yard should be added to the diameter of the pillar for every 6 feet of loose material between the surface and the firm strata beneath.

For the protection of surface erections around the shaft top, it is necessary to increase the dimensions of the pillar arrived at by this rule in proportion to the distance the buildings extend from the shaft. Assuming such erections are within a radius of 100 yards from the

shaft, it will be desirable to increase the size of the pillar by the same amount.

Shaft Sidings and Main Levels or Roads.—Before commencing to drive the main roads, it is necessary to decide their direction, and this in a great measure depends upon the shape and position of the coal royalties to be won and worked, and upon the dip of the seam. When the direction is decided and the main roads started, it should be borne in mind that the first 150 or 200 yards on each side will form the shaft siding. This is where the coals in the tubs or trams come to from every part of the mine to be sent to the surface, where all the empty tubs are dispatched into the mine, where all the men and boys travel when entering or leaving the mine, and through which all the air entering the mine must pass to go into its various parts or districts. It is therefore very desirable that this passage be made with plenty of width and height, so that the work may be carried on with dispatch, economy, and safety.

For 150 or 200 yards, therefore, from the shaft bottom, the roads made in the coal are of the requisite width to fulfil these conditions. Thereafter they are reduced to a width sufficient for ordinary haulage purposes. The height of the seam determines the first height of the road, and if it is not sufficient for the purposes required, part of the stone forming the roof or thill is removed.

In order to make the shaft siding secure, the roof and sides are very often supported by arching with brick or stone, or by building straight side walls and stretching iron or steel girders across from side to side to support the roof. In some cases only timber is used. Figs. 66 and 67 show a plan and section of an arched shaft bottom where the arrangements are adapted for a large daily output of mineral. It is very desirable to have shaft sidings perfectly straight. If the direction of the road has to be altered, or any branch roads started, they

should commence at the end of the siding, otherwise future complications may result.

The shaft siding having been formed, the levels are continued in the direction decided upon. If possible they should be driven with a slight rise from the shaft,

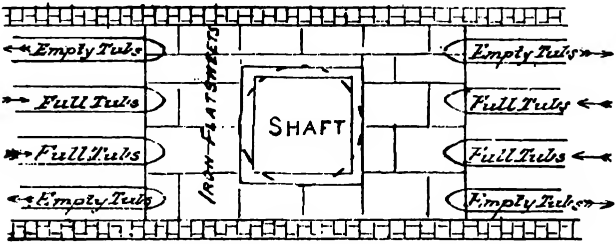


Fig. 66.—Plan of Shaft Bottom Arrangements

so that any water made in the workings will run to the sump. Sometimes special levels, termed *water levels*, are driven for the outflow of water when in large quantities, in order to keep the main roads, rolley-ways, or wagon-ways dry.

To facilitate the haulage of the loaded tubs or trams when moved by horses, the main roads should be given a slight rise from the shaft. Experiments have proved that a rise of about 1 in 130, or a little more than a quarter of an inch per yard, is the most advantageous gradient for horse haulage. A gradient of this amount may equalize the resistance of the loaded and empty trams, or very nearly so, the exact gradient for equality depending on the relative weights of the loaded and empty trams.

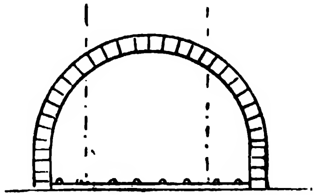


Fig. 67.—Section showing Arching to support Roof and Sides at Shaft Bottom

If the seam is not of sufficient height for a main road,

then the roof or thill is cut into according to circumstances, so as to make the height suitable. A main road is seldom made less than 6 feet, or more than 8 feet in height. If the seam much exceeds this in thickness, it may be advisable to leave part of the upper coal to form the roof. The width of the main roads is usually determined by the future requirements of the mine; they are seldom less than 6 feet or more than 12 feet.

Timbering of Main Roads.—It is very important that the sides and roof of main roads upon which men and boys travel, and which serve for the conveyance of

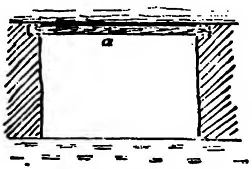


Fig. 68.—*a*, Crowntree supporting Roof

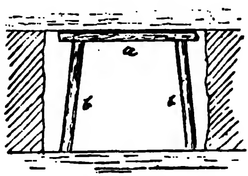


Fig. 69.—*a*, Crowntree. *b b*, Props supporting Roof

the coals to the shaft, and for the ventilation of the mine, should be properly secured to avoid falls of coal and stone. With a strong, framy sandstone roof, and substantial pillars of coal left on each side of the main roads, very little timber support may be needed; but with a shale roof, especially one tender and liable to crumble and break off short, and when there is much superincumbent weight, it is necessary to resort to supports.

The simplest method of supporting the roof only, is to let (or notch) into the coal or stone on each side, when it is sufficiently strong, a bar or crowntree (see fig. 68); or to notch into one side only, and place an upright prop at the other side. When the sides are not strong enough, or the span is too wide, a prop is fixed under each end of the crowntree slightly inclined from the perpendicular (see fig. 69). When the roof is very loose and crumbly, bars or polings are placed longitudinally upon the trans-

verse crowntrees, forming what is known as *lofting* or *covering* (see fig. 70). This prevents the loose stone dropping out between the transverse crowntrees.

In high and wide places, instead of crowntrees, heavier pieces of timber termed *balks*, and thicker props than usual, are used. Strong timber is used on all main roads, where there is horse or engine haulage, at sidings or pass-byes, and at all places where the nature of the roof requires it.

In wide places, where two sets of rails or a travelling road for men by the side of the rails is required, a more

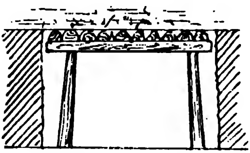


Fig. 70.—Method of Timbering Loose Stone

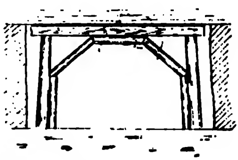


Fig. 71.—Method of Timbering where Strength is required

substantial system of timbering may be adopted. A balk is placed under the roof from side to side of the road, and a strong prop fixed under each end to support it; then a short prop is placed in a slanting position between the balk and prop, as shown in fig. 71.

Another method of timbering main roads, and one which gives great strength where there is much top and side pressure, is to run balks longitudinally along the centre of the roof of the roadway, and to place strong struts from them against the sides of the road. If the stone is loose, pieces of timber may be run longitudinally on the struts, the whole resembling the roofing of a house. When the sides are weak, these struts may be wedged tight against upright props placed against the sides.

The floor or thill of a seam is sometimes of a soft, yielding nature, and the upright props may sink into it when any weight comes upon them, and the floor rise

up in the middle of the road. When this happens, a piece of timber similar to that used on the top of the props must be laid upon the bottom and the props set upon it (see fig. 72).

Props should always be placed at right angles to the lines of the stratification, irrespective of the vertical line. The timber should be so arranged as to distribute the weight of the stone evenly along the length of the top piece. If the roof is not level, wedges and splits of wood should be inserted on the top of the timber in all places where the stone does not bear upon the timbers.

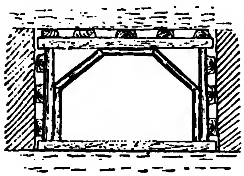


Fig. 72.—Method of Timbering where Roof, Sides, and Floor require strong Timber

The distance that each set of timbers is placed apart is determined by the nature of the roof and sides. With a strong roof the distances may vary, timbers being set up only where the stone shows signs of weakness, and contains joints and fissures. With a fairly good roof an average distance of 3 feet is a very frequent arrangement. When the roof is weak and rests heavily upon the timbers, it is necessary to place them closer together than this, and in some cases to have them almost touching.

Timber used in mines varies in size according to the nature of the stone and the width of the place. For main roads which are about 6 feet in height and 8 or 9 feet in width, the size of the top timber or balks is generally 6 or 8 inches square—sometimes the balks are split into two of 6 × 3 or 8 × 4 inches section, and used on main roads. The props used for supporting the balks are generally round, just as they have been cut, and with the bark left on; a useful size is 4 or 5 inches diameter. In wide places and where the roof is heavy, larger-sized timber may be required.

The kinds of timber most largely used in mines are pine, fir, larch, beech, and oak. The two former grow

straight with a regular diminishing section, and are well adapted for mining purposes, as they are elastic and tough, and bend very much before breaking. They are largely imported from Norway and Sweden. Beech timber is much used for chocks; oak is used mostly in shafts for buntons, cribs, pump collarings, &c.

The Preservation of Timber in Mines.—Timber in mines is subject not only to pressure from the roof and sides of the roads in which it is placed, but to rapid decay owing to the warm, humid atmosphere, particularly in the air roads, known as *returns*, along which the vitiated air travels from the workings to the upcast shaft. This decaying process is so rapid in some cases that the timber becomes quite rotten in a few months, and is then quite useless as a support. A renewal of timber thus becomes necessary. This is an expensive and may be a dangerous operation, to avoid which various methods have been tried to preserve the timber, in order to lengthen the period of its use underground. Its lifetime may be lengthened by having it cut down when there is little sap in it, and by using it only when well-seasoned. Such timber may last for many years in places where the ventilation is good, but otherwise it requires additional treatment. The methods are:

(*a*) Giving the timber a coating with whitewash.

(*b*) Painting it over with coal-tar. This is useful for places where the air is cold and moist.

(*c*) Steeping it in brine.

(*d*) Steeping it in solutions of iron or copper.

(*e*) Creosoting it. A thoroughly satisfactory process.

(*f*) Immersing the timber in boiling water to which has been added common salt and chloride of magnesium. This is known as the Aitken process.

(*g*) Subjecting the timber to an electric current. This is called the Nodon-Bretonneau process.

Arching Main Roads is adopted in mines where very loose strata and great pressure are encountered.

Shaft sidings are frequently arched, also main roads through faulted ground, and other weak places where timber would require frequent renewal. Arching is a very durable support, but its first cost is heavy. It should not be adopted where there is any likelihood of any movement of the strata, such as frequently occurs in mines by the floor heaving in the middle of the roads.

Fig. 73 illustrates the ordinary method of arching, the arch being usually built with ordinary fire-bricks, although in some cases dressed stone is used. When the floor is weak and soft, it may be necessary to construct an inverted arch upon which to rest the main arch. The thickness of the arch depends upon circumstances, but it is not less than 9 inches and rarely more than 27 inches. The side walls are usually built plumb, and the arch sprung from them. Very little mortar should be used in the joints, and all timber should be taken out and the space between the arch and the stone carefully packed. The packing sometimes consists of sand, which distributes the pressure evenly upon the arch.

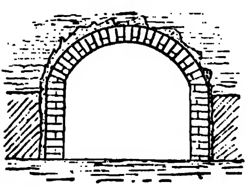


Fig. 73.—Underground Arch built with Bricks

Iron and Steel Supports for Main Roads.—Iron and steel girders and props are used in many places as substitutes for ordinary wood supports, and on shaft sidings and main roads they are found to be of advantage. Timber in most mines is subject to decay, becomes bent or broken by weight, and frequently requires renewing. Iron and steel supports are much more costly at first, but last very much longer than timber, and if they become bent or broken can be straightened or welded and used over again.

Plain iron or steel horizontal bars are sometimes used, supported by wooden props or resting on two side walls of masonry (see fig. 74). When the pressure is great

the bars are used bent into arch shape, and rest upon the masonry.

Rails, either disused or new, of wrought-iron or steel, are often adopted where great strength is needed, and may be applied either horizontally or bent into arch shape. Where the stone is strong on both sides the ends of the rails may rest in holes cut in the stone.

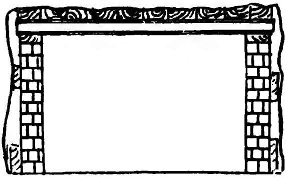


Fig. 74.—Steel Girder upon Brick Walls

When the rails are supported on wooden props, a broad flat piece of timber called a lid or headtree should be placed between the rail and the top of the prop to prevent the weight from forcing the rail into the latter and splitting it. Old tramway rails make excellent supports.

A method of applying iron rails, adopted in some Continental mines, is illustrated in fig. 75, and is fitted to resist great top and side pressure. The rails are bent into the required shape, and fastened together by means of ordinary fish-plates and screw-bolts. They are placed at short distances apart, and wooden planks are stretched from one set to the other to fill up the space, and so prevent any pieces of stone falling out from between the sets.

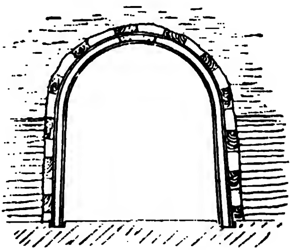


Fig. 75.—Illustrating method of using Iron or Steel Rails to support Roof and Sides

Girders are generally made of double T-headed section, thus I.

Driving Levels through Stone.—Levels are not always made through coal. They are often required to

go through stone, as, for example, when a fault has been met with, a drift is driven in order to cut the coal on the faulted side; or sometimes it is required to make an inclined communication from one coal-seam to another by cutting through the intervening strata. Such drifts are termed *stone drifts*. They may be driven level, rising, or dipping, and at such an angle of inclination as may suit the circumstances of the case. Thus, if the drift is to be used for a haulage road an easy gradient will be adopted, and if for ventilative purposes only, a sharp gradient may be used to make the drift as short as possible.

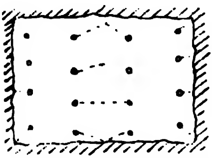


Fig. 76.—Face of Stone Drift showing the arrangement of Shot-holes

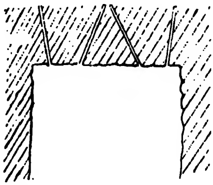


Fig. 77.—Plan of Face of Drift and Shot-holes

The size of stone drifts depends upon the purpose for which they are required.

Blasting operations are usually required in driving stone drifts. The placing of the shot-holes at the face is carried out in a manner similar to that described in the sinking of shafts through stone. The centre and bottom are usually unkeyed first by a heavy explosive charge, after which the other parts of the drifts are more easily blasted. The sides and top of the drift should be properly dressed and squared, ready for the timbering, and no overhanging stones left (see figs. 76, 77). The boring of the holes is sometimes done by hand with ordinary drilling tools or by ordinary hand-drilling machines. In long drifts through hard stone, machine-drills worked by compressed air are often adopted. There is usually much scope for the exercise of skill, and knowledge of the principles of blasting, in the

placing of the holes to do the greatest amount of work in stone drifts.

How to drive a level in a given direction and to a given inclination.—A level may be driven in a straight line and in any given direction by means of plumb-lines suspended from the roof. A compass is set in the drift, usually in the centre, and the magnetic needle allowed to come to rest. Then the head of the instrument is turned in the exact direction which it is intended to drift. Two or three plumb-lines are then suspended from the roof at intervals of 3 or 4 feet in the line of the direction of the drift, and by the help of these lines the centre of the face of the drift may be kept in the direction required, as follows: the observer stations himself behind the plumb-lines, and another person stands at the face of the drift with a light; as soon as the suspended lines come to rest the observer directs the person to move the light until it is in the line of the plumbs, which is, of course, the line of the direction of the drift, and the position of the light should be the centre of the drift. Where the bearing of the dip and rise is known, and it is desired to drive a level or cross entry approximately along the strike or level course, but having a slight rise in its favour to facilitate haulage and drainage, the direction may be found as follows:

Let A = Angle between the proposed road and the strike of seam.
x = gradient of proposed road in degrees.
y = gradient of seam in degrees.
z = percentage gradient of proposed road = $\tan x$.

Then $\sin A = \dfrac{\tan x}{\tan y} = \dfrac{z}{\tan y}$.

EXAMPLE.—A coal-seam dips 10 degrees along a bearing N. 30° 45′ E. Two cross entries are to be set away one from each side of a dip brow. If the gradient in each case has to be 1 per cent rise, find the bearing of each entry?

$\text{Sin}\, A = \dfrac{z}{\tan y} = \dfrac{0{\cdot}1}{\tan 10^\circ} = \dfrac{{\cdot}01}{{\cdot}1763} = {\cdot}0567 = \sin 3^\circ\, 15'$.

But strike is at right angles to dip, ∴ bearing of strike is

S. 59° 15′ E., and bearing of entry to right is (S. 59° 15′ E.) − (3° 15′) = S. 56° E. Bearing of entry to left is (N. 59° 15′ W.) + (3° 15′) = N. 62° 30′ W.

A drift may be driven to any gradient by means of an ordinary spirit-level, or by a wood-level or plumb-bob (see fig. 78). The latter is made of wood, and consists of one horizontal piece about 3 feet long, and a vertical piece about 2 feet, with a plumb-line hanging in the centre from the top when it is level. When it is not standing level the plumb-line does not hang in the centre. When used in an inclined drift the horizontal or foot piece is set at an angle with the vertical piece corresponding to the angle of inclination required in the drift; or a piece of wood is added to it at one end equal to the amount of inclination. Thus, if the inclination is 1 inch per yard, then a piece of wood 1 inch thick nailed on to the underside at one end of the foot-piece will be required. It is used by placing it upon the tram-rails, and if the plumb-line hangs in the centre the inclination of the drift is correct.

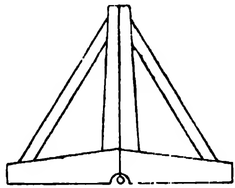


Fig. 78.—Wood-level or Plumb-bob

CHAPTER IX

TOOLS USED IN COAL MINING

Various forms of Picks, Wedges, Hammers, and Shovels used in Mines. Hand-boring Gear for making Shot-holes. Sharpening and Tempering Tools.

Picks.—These are the most ancient and most largely used of all mining tools. Rough specimens of picks, made of wood in some cases, similar in construction to

those used at the present time, have been found in caves and very old mine workings.

A pick consists of a blade or head, made of iron or steel, and a shaft or handle of wood. The blade varies in weight from 2 to 8 lb., according to the nature of the work to which it is to be applied. In the middle of the blade is a hole or eye, into which the shaft is fitted, and from the eye the blade, which is generally of a square pyramid form, tapers on each side to a point. The amount of taper depends upon the hardness of the coal or stone upon which it has to be used. For instance, if required for work in hard stone the taper is short, if in soft stone or coal the taper is made longer and thinner. The blade is sometimes made so as to be square or at right angles to the shaft. In this form it is adapted for work in places where a long reach is required, and for getting into nooks and corners. Sometimes the blade is curved inwards to the shaft; this form is adapted for underhand or bottom hacking work. A common form is the anchored, in which the blade on each side of the eye is straight, but of anchor shape—that is, makes an angle less than a right angle with the shaft.

The handle or shaft of the pick is usually made from 30 inches to 36 inches in length, and of an oval section suitable for a man's grasp. Where it is fitted into the eye of the blade its section is made a little larger, and it is firmly secured to the latter by iron wedges. In this way, one blade and one shaft make up a pick.

With *Hardy's pick* for hewing coal one shaft serves for a number of blades. It is known as the *interchangeable*, because when one blade becomes blunt it can be taken from the shaft and another blade put on by the hewer in his place of working. The interchangeable pick is also more convenient in the process of sharpening the blade. With an ordinary pick the wood shaft is exposed to the heat of the fire during the heating of the blade, whereas with the interchangeable the blade is detached from the shaft before being put into the fire.

Coal Picks.—Picks used in hewing coal are made very much lighter, and with much more taper than those used for stone work, such as shaft sinking or stone drifting. Fig. 79 represents a pick used for holing or undercutting the coal where a long reach is required. The blade is straight, about 18 inches in length from tip to tip, and weighs about 2 lb. The shaft is made from 3 feet to 4 feet in length. Fig. 80 shows the *interchangeable* as now made. To strengthen the shaft, and also to facilitate the putting on and taking off of the blade, an iron ferrule or hoop is affixed, as shown in the figure. The weight of the blades varies

Fig. 79.—Coal Pick used for "holing" or "undercutting"

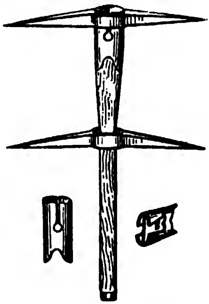


Fig. 80.—Interchangeable Pick, largely used by coal miners

from $1\frac{1}{2}$ lb. to 3 lb. The shafts are about 30 inches in length.

Stone Picks.—These are used upon rock only, and require to be made much stronger and heavier than those used upon coal, and the points are usually made bluff. Fig. 81 represents a stone pick as used in the North of England. The blade is slightly anchored, and the angles are chamfered so as to give an octagonal section; sometimes, however, the section is square. The blade is 20 inches and the shaft 30 inches in length, and the total weight is $7\frac{1}{2}$ lb. Fig. 82 represents a *bottom pick*; that is, one used for underhand work, as in cutting the floor of a seam—taking up bottom stone, as it is frequently termed. The blade is 21 inches, and

the shaft 33 inches in length. One of the points is made wedge shape, and the other terminates in a chisel edge.

Wedges.—These are used in mines to force off pieces of coal or stone where blasting is undesirable. They

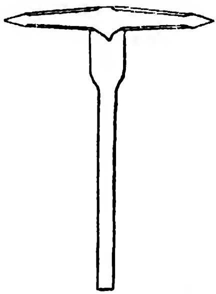


Fig. 81.—Stone Pick, used in sinking, stone drifts, &c.

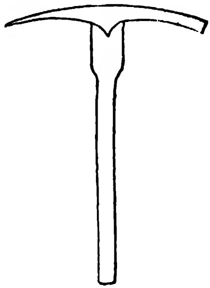


Fig. 82.—Stone Pick, used in cutting bottom stone

may be applied by driving them into a *parting*, a joint, or into a bore-hole drilled for the purpose. Wedges are made of steel, or iron with steel edges. Their length varies from 6 inches to 12 inches, their width from 1 inch to $2\frac{1}{2}$ inches, and their thickness from $\frac{1}{2}$ inch to $1\frac{1}{2}$ inches. Those to be used in coal are generally made much larger than those to be used in stone. Fig. 83 represents a coal wedge, and fig. 84 a stone wedge.

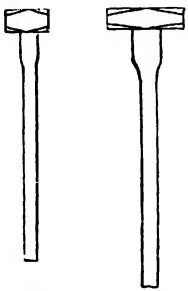
Fig. 83.—Two views of a Coal Wedge

Fig. 84.—Two views of a Stone Wedge

These are examples of the simplest kind of wedges. For other kinds see Chapter X on Mechanical Substitutes for Blasting in Fiery Mines. The length of the coal wedge is 12 inches, and at the base of the wedge the section is $2\frac{1}{2}$ inches broad by 1 inch thick, and tapers 6 inches to the edge. The striking face is octa-

gonal. The weight of this wedge is 7 lb. The stone wedge is about 6 inches long, and at the base the section is $1\frac{1}{2}$ inches broad by 1 inch thick. Its weight is 2 lb.

Hammers.—These are sometimes termed *sledges* and *malls*. They are used in mines for various purposes, such as for driving wedges into coal and stone, for breaking large pieces of stone, for striking drills in boring shot-holes, and for forcing props up tight against the roof. There is not in hammers much variety of form and dimensions; those used for driving wedges are slightly different from those used for striking drills. Of the latter there are the single-handed, for use in one hand; and the double-handed, for use in both hands.



Figs. 85, 86.—Hammers used in coal mining

The head of a hammer is usually made of steel, the striking ends or faces being hardened. In the centre of the head is a hole or eye for the handle or shaft. The head is made from 4 inches to 9 inches in length, of 2 inches to 3 inches square section, with the edges bevelled or chamfered to reduce the weight, and also in striking to avoid injuring the hands of the person holding the drill. The weight of the heads varies from 5 lb. to 10 lb. The shaft or handle is generally made of ash or hickory. Fig. 85 is a sketch of a hammer for general work. The head is 4 inches long, octagonal section, edges chamfered, weight 6 lb. The shaft or handle is 24 inches long. Fig. 86 is a hammer for wedge driving. The head is 8 inches long, shaft 27 inches long, and total weight 10 lb.

Shovels.—The pick, wedge, and hammer are used for breaking down and loosening masses of coal and stone; the shovel is used for collecting and throwing

or shovelling the coal or stone into tubs, which convey the material to the surface.

A shovel consists of an iron or steel plate, or shovel proper, and a handle of wood. The plate is made slightly concave and circular, except the front edges, which are brought to a point. The sides are shouldered or turned up slightly so as to hold the material on the plate. The shovel for shovelling coal is made larger than that for stone. Fig. 87 represents a cast-steel socket pan shovel, size 16 inches, used for coal. The shovel of the same pattern for stone is usually made about 12 inches in width.

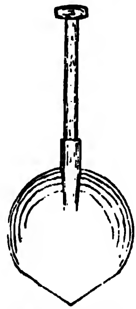


Fig. 87.—Coal Shovel

Drills or Borers.—Frequently coal and stone are so strong and tenacious as to render the work of breaking them down a very slow and laborious process. Explosives are then applied to do the work more cheaply and expeditiously. Explosives are used in shaft sinking, stone drifting, and in coal getting, and their application is termed *blasting*. Long, narrow holes are drilled or bored in the rock, into which the explosive is inserted. When this is fired, the force of the explosion dislodges some of the rock.

Drills or borers are the most important tools used in blasting operations, as they are used to make the holes in the rock in which the explosive charges are placed. A drill consists of a steel rod of octagonal section with a bit or chisel edge at one end, and a striking face at the other end. The cutting or chisel edge of the drill may be made straight or slightly curved; the latter shape is more common, as it is less liable to have the corners broken off in the hole. The angle of the cutting edge varies with the nature of the material it is to be used upon; thus it is made much less obtuse for drilling in coal than for stone. The width of the cutting edge

depends upon the size of the hole required, but is seldom less than 1 inch or more than 2 inches.

Drills are made in lengths varying from 20 inches to 42 inches. What is termed a *single-handed* set of blasting gear is used by one man, who holds a drill in one hand and strikes it with a hammer wielded by his other hand. The set consists of a short drill about 20 inches, a long drill 42 inches, a hammer with a handle about 9 inches and a head 7 inches in length. A *double-handed* set of blasting gear is used when two or more men do the work. One of the men holds the drill and the others strike it. The set consists of a short drill 18 or 20 inches long, middle drill 27 inches long, and long drill 42 inches; also a hammer with a handle 30 inches and head 8 inches long. The single-handed set is generally used in coal of soft shale, or in corners where there is little room; the double-handed set may be used for drilling in hard stone in stone drifts and in sinking pits. (See fig. 88.)

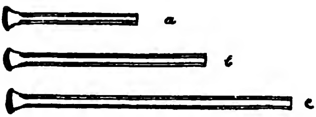


Fig. 88.—Set of Drills or Borers

a, Short drill. *b*, Middle drill. *c*, Long drill.

The operation of drilling is very simple. A man holds the drill in one or both hands, and blows are dealt on the striking face by a hammer, which forces the cutting edge into the stone. After each blow the drill is drawn outwards a few inches and turned slightly, so as to form a circular hole. When the short drill has bored its length, the longer drill is then substituted.

A drill, called a *jumper* or more frequently a *churner*, is sometimes used in boring a hole. This drill does not require the force of a hammer. It is generally made from 4 feet to 6 feet in length, with a chisel edge at each end; half-way from each end the steel is bulged out to form a heavy ball, in order to give the drill greater weight. To use it, a short hole is first made with the

pick, then the drill is inserted, and at each stroke it is *jumped* or pushed sharply into the hole, and in this way the hole is drilled.

Hand-boring Machines.—Hand-boring machines with a rotary motion are much used in mines nowadays, instead of the sets of drilling gear just described. They bore the holes at a much greater speed, and require less manual labour. They are made very light for drilling in coal, and may be worked by one man.

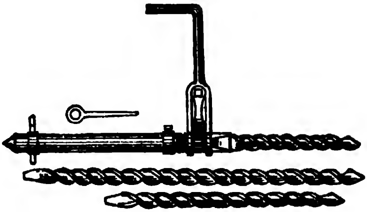


Fig. 89

Usually a machine consists of an iron frame called a *standard*, which may be fixed between roof and thill, or between any two other bearing surfaces, at a convenient distance from the face. It is usually made in telescopic fashion so as to be adjustable to various heights of roadway, and can be easily set and screwed up so as to remain quite rigid during the process of boring. A screwed rod passes through an iron box, which is threaded to suit the rod, and is also free to slide up and down the standard to suit the position of the hole. A handle or ratchet is attached at one end of the screwed rod, and twisted or auger-shaped

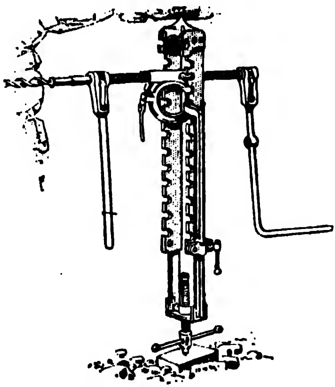


Fig. 90.—The Elliott Boring Machine

drills at the other end to bore the hole (see fig. 90). A backward-and-forward movement given to the ratchet by the man causes the auger to revolve, and the great amount of power gained by the screw causes it to bore out the material in contact with it. In fact, boring-machines are made to penetrate the very hardest of stone, and may be fixed to bore at any required angle (see fig. 90).

Machine Rock-Drills.—For quick drilling of holes in shaft sinkings, in ironstone mines, and stone drifts, machine rock drills, worked by steam, compressed air, or electricity, are being much used. They are made to do the work of drilling percussively, that is, in the same way as the ordinary hand drills with hammer. They are provided with a cylinder, piston, and piston-rod, to which the cutting tool is attached. Arrangements are made in their construction to cause the drill to rotate slightly at each stroke, so as to bore a circular hole, and to give it a forward or feed motion. An example is the "Hardy Simplex" Drill. It weighs 28 lb., drills to a depth of 6 feet easily, and actual tests show a speed of 8 inches per minute in hard sandstone up to 24 inches per minute in coal.

Sharpening and Tempering Tools.—The sharpening and tempering of picks and drills is a work requiring much care and attention. Each coal hewer may blunt three or four picks per day, and, in addition, there are always a number of stone picks and drills which require to be sharpened and tempered.

The picks are heated in the smith's fire, and the points hammered and sharpened with a small hammer on an anvil. During use in the mine, the points of the pick become worn or blunted, broken off or turned up. When heated, they are carefully drawn out on the anvil, and then subjected to the process of tempering.

Stone picks usually become much more damaged than coal picks, and require more care in sharpening. The

smith should be acquainted with the nature of the stone upon which the tools are to be used, in order that he may harden and temper them to suit.

Ordinary hand drills are sharpened by being beaten on an anvil. The cutting edge becomes jagged, turned up, broken or worn by use, and to get it again into the required shape and angle it is heated to bright redness and beaten on an anvil, then finally dressed with a file.

The process of tempering follows that of sharpening, and requires much care. Steel becomes hardened when subject to a rapid reduction of temperature; as, for instance, when it is heated and then suddenly plunged into water, oil, &c. The more rapid the reduction in temperature, the harder will the steel become. In fact, it can be rendered hard enough to cut glass. If allowed to cool gradually after being heated to a red heat, the steel will be as soft as iron. When steel has been sharpened as required, it has to be tempered to suit its work; this is done by reheating it, when it will show various colours, which appear in the following order, and from which the various degrees of hardness can be obtained. These show the change from brittleness to softness: (*a*) pale yellow or light straw colour; (*b*) dark straw colour; (*c*) gold colour; (*d*) brown or brown and purple mingled; (*e*) purple; (*f*) violet; and (*g*) deep blue.

The degree of hardness best suited for the tools to work upon the different kinds of stone, is to a large extent found by trial, the smith having to use his own judgment when he sees the effect of the stone upon the tools.

To temper picks they are reheated a little after sharpening, and then dipped a few inches into water and moved a little up and down. The smith at the same time watches the tints as they appear, and as soon as the proper one is observed, suddenly plunges the tool completely into the water and cools it off. When withdrawn it will be of the required hardness.

Drills are tempered in the same way. Twisted or auger drills are sharpened by heating and filing.

CHAPTER X

BLASTING AND EXPLOSIVES

Cleaning, Loading, and Firing Shot-holes. Fuses and Detonators. Electric Blasting. Composition of Explosives. Safety Regulations for Blasting. Mechanical Substitutes for Blasting in Fiery Mines.

Cleaning Shot-holes.—Before loading a shot-hole with an explosive charge, it must be very carefully cleaned of any dust or sludge that may be in it, which has been made during boring. The tool employed for this purpose is called a *scraper*. It consists of a copper rod about $\frac{1}{4}$ of an inch diameter, and usually about 4 feet long. The end of the rod which enters the shot-hole is flattened into a circular form of less diameter than the hole, and turned up at right angles to the rod. To clean a shot-hole, this instrument is pushed to the bottom of it and then withdrawn, bringing with it the dust; this operation is repeated until the hole is quite clean and ready for an explosive charge. (See fig. 91.)

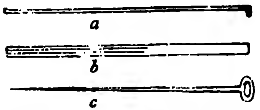


Fig. 91.—*a*, Scraper. *b*, Stemmer. *c*, Pricker used in Blasting

Charging and Firing.—The explosive charge to be used must be enclosed in a case or cartridge, which must be about the same diameter as the hole. It is put into the hole, and gently pushed to the end or bottom by means of a tamping-rod or stemmer made of copper. When the charge of gunpowder is to be fired with a straw, before putting it into the hole a copper pricker (which is a rod of copper of small diameter terminating

in a point) is inserted into it, and by means of it the charge is pushed to the end of the hole. The pricker is left sticking in the charge during the tamping or stemming of the shot-hole. Tamping is necessary in order that the full explosive force may be exerted upon the rock, which would otherwise spend itself along the hole without dislodging any stone. A shot-hole may be 36 inches long, and the charge only 6 inches; this leaves 30 inches of bore-hole to be filled up or tamped. The material used for this purpose must not be coal, but may consist of clay, soft shale, broken bricks, or similar substance.

The tamping-rod or stemmer for forcing this material in tight is made of copper, or phosphor bronze. It is usually about 3 feet long, and of semicircular form, with a small groove along it for the pricker or fuse, which communicates from the charge to the mouth of the hole. When the tamping is completed, the pricker is slowly and carefully withdrawn, leaving a narrow hole through the tamping into the charge.

To fire the shot, a straw or squib containing a train of powder is inserted into the hole, with a piece of touch paper attached to the outer end. The touch paper is lighted, and when it burns to the straw it ignites the powder, and this in its turn fires the charge.

This method of firing shots is very simple, but is only permissible in certain mines and then only when certain safeguards are adopted.

Fuse is made with a central core of grains of powder, surrounded with cotton, and covered on the outside with gutta percha to make it waterproof. The end of the fuse is put into the charge and secured to it, before the latter is inserted into the hole. The charge is put to the end of the hole, and the other end of the fuse is left projecting a safe distance from the mouth of the hole. The hole is then stemmed, and the fuse ignited. This may be done by means of an igniter contained in a tube which, when attached to the fuse, forms a com-

pletely closed chamber, or in naked light mines by means of a lamp or candle. Fuse is necessary in wet places, as, being waterproof, the wet cannot affect the train of powder. Fuse burns at the rate of about one yard in one and a half minutes, and, knowing this, the person in charge can calculate the length necessary to enable him to reach a place of safety before the charge explodes.

Electric Blasting.—Electricity is now much employed in blasting operations in mines, and its use is attended with many advantages. In shaft sinking, stone drifting, or tunnelling, saving of time is effected by firing a large number of shots simultaneously instead of firing them one by one; and this can easily be done by electricity. When electricity is used, what is known as a " miss-fire " or failure to explode is not frequent, but when it does occur, there is no danger in immediately approaching the shot-hole directly the wires are disconnected from the electric machine. Whereas with a fuse, if the hole does not explode at the proper time, it is necessary to wait some time before approaching it, as it might be " hanging fire ", and explode unexpectedly. The Explosives Order stipulates that in the case of a miss-fire ten minutes must elapse before approaching shots electrically fired, and one hour in the case of shots fired by other means.

It has been proved by experiment that a much greater explosive effect is produced by electric blasting than by the other method. Another great advantage is that no sparks are produced, except in the interior of the charge.

For electric blasting the following are required:

(*a*) An exploder.
(*b*) Firing cable.
(*c*) Detonator and wires.
(*d*) Explosive charge in shot-hole.

Exploders.—These may be high or low tension. They are alike in construction and principle, with the excep-

tion that the high-tension machines are wound to give a small current at a pressure of from 50 to 100 volts, while the low-tension give a greater current at a lower voltage.

Magneto-electric exploders have now superseded batteries for firing the charge in underground blasting.

Firing Cable.—This consists of two insulated wires. For shaft sinking the wires are usually separate, but for underground purposes two wires are usually laid together to form a twin cable. The insulation usually consists of tape or tarred flax drawn through composition. The cable ought to be of sufficient length to allow the shot-firer to be in a place of safety when he fires the shot.

Detonators.—These are of two kinds, high-tension and low-tension.

The high-tension is a small copper tube closed at one end, and partly filled with an explosive compound consisting of 95 per cent of fulminate of mercury and 5 per cent of chlorate of potash. Wires are fixed in the tube, and in firing a spark is produced between the ends of the wires embedded in the explosive mixture, and causes a sharp explosion.

The low-tension detonator is similarly constructed, but the heat to produce the explosion is generated in a small loop of high-resistance wire embedded in the mixture.

As all high explosives require a sharp initial explosion in order to detonate them, detonators are necessary for this purpose.

Explosive Charge in Shot-hole.—The explosive is made up in cartridges, usually of 2 or 4 ounces in weight, covered with paper. They should be of suitable diameter to suit the size of the shot-hole, because it is necessary for effective blasting that the cartridges should fit the hole. The cartridge should be ½ inch less in diameter than the shot-hole to avoid any pressure upon the cartridge. If a cartridge is too small, by slitting it a little in the middle, and pressing it when in the hole

with the stemmer, it may be expanded to fill it. The number of cartridges required to make up an explosive charge depends upon the amount of work the shot is required to do, and is determined by the experience of the person in charge.

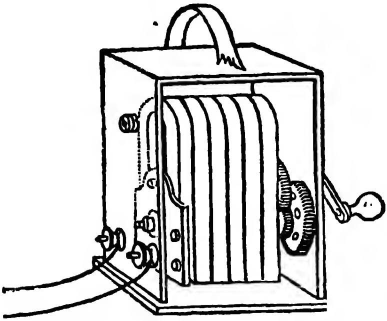


Fig. 92.—Magneto-electric Exploder

The charge having been put into the hole, with the detonator inserted in one of the cartridges, the hole is then tamped or filled up with clay gently pressed down upon the cartridges.

The ends of the fuse wires, which are left protruding from the hole, are separated and scraped clean, and connected with the ends of the firing cable by twisting them together: care must be taken that the two wires (lead and return) do not touch

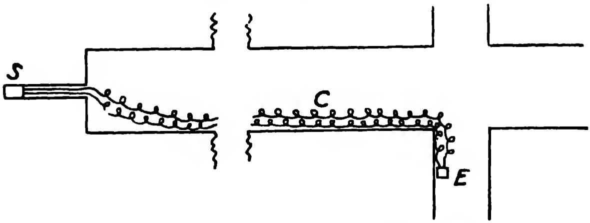


Fig. 93.—Electric Shot-firing in working place

S, Shot consisting of cartridges and detonator. C, Cable. E, Electric Exploder and position of Shot-firer.

each other. The free ends of the firing cable should be carried to a place where the operator will be safe from flying fragments of stone or coal, and there connected up to the terminals of the exploder (fig. 92) as shown in fig. 93.

When about to fire the charge, the magneto exploder is steadied with the left hand. The driving handle of the exploder is then turned sharply and contact made by pressing in the switch button, when the current passing along the wires explodes the detonator, the shock from which in turn explodes the charge.

It is important that no connection be made between the cable and exploder until everything is ready, everybody away from the vicinity of the shot, and it is intended to fire immediately.

Immediately after firing a shot, the shot-firer should disconnect the cable, then disengage it from any coals or stones that may have fallen upon it, and coil it up.

For firing several shots in series or circuit, a sufficiently strong exploder is required. The fuse wires are connected so that one of the wires of the first shot, taken in the order of the series in circuit, is connected to one of the wires of the second shot, and the other wire of the second shot to one of the wires of the third shot, and so on in succession until the whole series are connected. Each end wire is connected to the wires of the cable, the other ends of which are attached to the exploder, and all the shots are fired simultaneously.

Another method of firing several shots at once is to fire in direct contact or in parallel. One wire of each fuse is connected direct to one end of one of the cable wires, and the other wire of each fuse to one end of the other cable wire, and in this way all shots are fired simultaneously.

Composition and Properties of Various Explosives.

GUNPOWDER.—This explosive has been most extensively used in blasting operations in mines, but owing to the great danger attending its use, governmental restrictions have been imposed, and it is very much less frequently applied now than formerly. It damages coal less than other explosives owing to its slow combustion, which produces a rending action, and where it can be used without danger, it is usually adopted in

preference to the higher and stronger class of explosives, which exert a shattering force, and break up the coal very much. Its composition by weight is: nitre, 75 per cent; charcoal, 13 per cent; sulphur, 12 per cent. These proportions may be varied, according to the class of work for which it is required.

Gunpowder is comparatively safe to handle and transport, is easily manufactured, and is cheap. On the other hand, it is easily rendered useless by moisture, and its fumes are of a noxious character. Its use in mines, generating firedamp and producing much coal dust, has been strongly condemned, owing to the large outburst of flame which usually attends the explosion, and which may ignite the firedamp and coal dust.

NITRO-GLYCERINE.—This is a liquid explosive of great power, but it is not used in mining operations in its liquid state on account of the great danger attending its handling, carriage, and storage. Many serious accidents have occurred owing to its explosion from slight concussion. It is a clear, colourless, or yellowish oil, of sweetish and burning taste, of 1·6 specific gravity. A very small quantity of it taken into the human system will produce fatal results.

DYNAMITE.—Owing to nitro-glycerine being so dangerous in its liquid state, various experiments were made with solid absorbent substances, and it was found that a porous substance, a silicious infusorial earth called *Kiesulguhr*, found in Hanover, would absorb the nitro-glycerine, and while not reducing its explosive power, render it safe for handling, transport, and storage. Dynamite is therefore nitro-glycerine absorbed by Kiesulguhr, which takes up from three to four times its own weight of nitro-glycerine. It is of a pale reddish-brown colour, and is of a pasty or plastic nature, and is prepared in cartridges from 4 to 8 inches in length, and wrapped in parchment paper. When thrown into the fire it burns quickly; if struck sharply or subjected to any severe concussion it may explode.

In blasting with dynamite, fuse and a detonating cap are used. A detonating cap or detonator is a small metallic capsule or cap containing fulminate of mercury. When the charge of dynamite has been put into the hole, a primer is placed on it, consisting of a detonator with the fuse pinched into it embedded in a short cartridge of dynamite. To insert the cap and fuse into this short cartridge, the latter must be opened at one end and a hole made into it with a pointed stick; the cap is then put full length into the hole, and the explosive substance pressed around it, then the cartridge paper is gathered and tied around the fuse. It is then lowered into the hole and the tamping proceeded with, soft clay being used for the first few rounds.

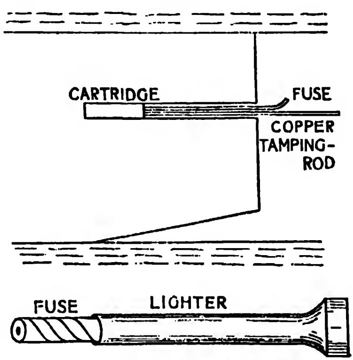


Fig. 94.—Charging and Stemming of Hole to be fired with Fuse and Bickford's Safety Lighter

In deep holes water is an excellent tamping, but where there is not much weight of water it is necessary to tamp with solid material. Dynamite is useful for blasting in wet places, as in sinking shafts, as water does not injure it or reduce its explosive force, and it is estimated to be from five to ten times more powerful than powder. Moreover, the fumes given off by an explosion are not more noxious than those from powder. Dynamite freezes more readily than water, is difficult to explode when in that condition, and there is danger in thawing it. It should be kept in a dry place at a temperature between 50° and 60° F. to avoid this.

"Flameless" or "Permitted" Explosives.—The frequency of colliery explosions has drawn considerable attention during recent years to the danger of blasting

underground, especially on main haulage roads, with explosives which produce flames. Endeavours have been made to discover some compound which would do the work of blasting without any flame whatever being produced by the explosion. Several explosives have been invented claiming this property, and in consequence have been termed " flameless explosives ", but it has been found that none are absolutely flameless. In " The Explosives in Coal Mines Order of the 21st May, 1912, regulating the supply, use, and storage of explosives ", the circumstances under which only " flameless " or " permitted " explosives are to be used are set forth, and also a list of the permitted explosives and their chemical constituents is given. The following are taken from the list, which is subject to revision in accordance with the results of experiments which are made from time to time in the Government Testing Stations:

AMMONITE No. 2, consisting of the following mixture:—

Nitrate of ammonium	75 parts
Nitrate of potassium	18 ,,
Di-nitro-naphthalene	3 ,,
Chloride of sodium	8 ,,
Moisture	1 ,,

To be fired with a detonator of not less strength than that known as No. 6.

BELLITE No. 2, consisting of the following mixtures:—

Nitrate of ammonium	62·5 parts
Di-nitro-benzol	13 ,,
Chloride of sodium	28·5 ,,
Moisture	·75 ,,

To be fired with an electric detonator No. 7

CAMBRITE, consisting of the following mixture:—

Nitro-glycerine	24 parts
Nitrate of barium	4·5 ,,
Nitrate of potassium	29 ,,
Woodmeal	35 ,,
Carbonate of calcium	·5 ,,
Oxolate of ammonium	9 ,,
Moisture	6 ,,

To be fired with No. 6 detonator

DYNOBEL, consisting of the following mixture:—

Nitro-glycerine	33·5	parts
Nitro-cotton	1	,,
Perchlorate of potassium	28	,,
Woodmeal	10·5	,,
Oxolate of ammonium	30·5	,,
Moisture	1·5	,,

To be fired with No. 6 detonator

ESSEX POWDER, consisting of the following mixture:—

Nitro-glycerine	24	parts
Nitro-cotton	1·5	,,
Nitrate of potassium	35	,,
Wheat flour	35	,,
Chloride of ammonium	7	,,
Moisture	5	,,

To be fired with No. 6 detonator

KENTITE, consisting of the following mixture:—

Nitrate of ammonium	35·5	parts
Nitrate of potassium	35	,,
Chloride of ammonium	18	,,
Tri-nitro-toluol	16	,,
Moisture	2	,,

To be fired with No. 6 detonator

PIT-ITE NO. 2, consisting of the following mixture:—

Nitro-glycerine	25	parts
Nitrate of potassium	31	,,
Woodmeal	36	,,
Oxolate of ammonium	9	,,
Moisture	5	,,

To be fired with No. 6 detonator

UPLEES POWDER, consisting of the following mixture:—

Nitrate of ammonium	65	parts
Tri-nitro-toluol	6	,,
Chloride of ammonium	15	,,
Nitrate of sodium	14·5	,,
Starch	4	,,
Moisture	1·5	,,

To be fired with No. 7 detonator

WESTFALITE No. 3, consisting of the following mixture:—

Nitrate of ammonium	61 parts
Nitrate of potassium	15 ,,
Chloride of ammonium	22 ,,
Tri-nitro-toluol	6 ,,
Moisture	1 ,,

To be fired with No. 6 detonator

In each case the regulations require that in addition to the marking on the outward package required by an Order of the Secretary of State made under the Explosives Act, 1875, and in force for the time being, such outer package shall bear the words "As defined in the List of Permitted Explosives", and, further, that each package shall be clearly marked with the words "Permitted Explosive to be used with not less than No. 6 or No. 7 Detonator", as the case may be, and also with the name of the explosive, the name of the manufacturer, the date and place of manufacture, and the nature and proportion of the ingredients.

Explosives enclosed in Flame Extinguishers.—Attempts have been made to produce flameless cartridge cases to extinguish any flame produced when the explosive ignites, but these are not permissible.

Safety Regulations in Blasting.—All the rules of the Coal Mines Act, 1911, and of the Explosives in Coal Mines Order relating to blasting operations in mines should be strictly observed. With regard to the transport and storage of explosives, the provisions of the Explosives Act, 1875, should be adhered to, and also any instructions issued by the makers of the explosives used.

Many accidents have occurred from the use of straws, which sometimes explode the charge before the person who lighted the touch paper has had time to reach a place of safety. Then sometimes the shot "hangs fire", that is, does not explode at the time anticipated, owing to the slow burning of the train of powder in the straw, or some other cause. In such a case great risk attends

the person approaching the shot again from sudden explosion. The use of ordinary fuse avoids the first kind of accidents, but not the second, as it is liable to " hang fire " as well as the straw. There is also danger in the exposure of a naked light by unscrewing a safety lamp in a fiery mine to light the straw or fuse, and although a wire heated by passing it through the gauze of a safety lamp to the flame to ignite the fuse has been used, both methods are now prohibited.

Electric firing obviates all these dangers. By its adoption there is no risk to any person in firing, as everybody can be in a place of safety before the wires are connected to the machine. There is no risk of " hanging fire ", and the shot may be approached as soon as the leading wires are disconnected from the electric machine if it does not explode with the current. Lamps need not be unlocked, and no sparks are produced by the machine or wires except in the shot-hole.

As already mentioned, a great danger in the use of explosives in mines arises from the flame which generally attends the explosion of a shot tamped or stemmed in the ordinary manner, and which issues in large volumes from the mouth of a hole which has been overcharged, or when the coal or stone resists the shot and as a result the tamping is suddenly blown out of the hole. This is termed a " blown-out shot ". Firedamp and coal dust may be ignited from these causes, and give rise to a serious explosion. The use of powder is more likely to result in these dangers, and therefore its use as a blasting agent has been strongly condemned, and prohibited in gassy mines. Only permitted explosives, fired with fuse or electricity, are allowed to be used, to secure reasonable safety from danger in gaseous and dusty mines where blasting is needed.

Mechanical Substitutes for Blasting in Fiery Mines.

Wedges.—Ordinary wedges, already described, are sometimes used to bring down coal in mines, but are

unsuitable for stone work, except for wedging off small pieces which may have been left by a shot.

Wedge and Feather.—This consists of three iron wedges. To use them a bore-hole is drilled in the coal or stone to be operated upon, and two of the wedges are placed in it with the thin end outwards, then the third wedge is driven between them (see fig. 95).

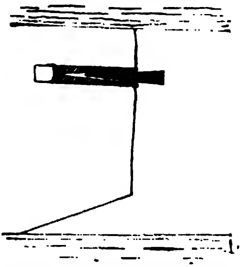


Fig. 95.—Wedge and Feather applied to force down Coal

Patent Multiple Wedge.—This consists of five parts, which are applied as follows: A hole is drilled of the required size and length, and two wedges, one above the other, are placed in the hole with the thin edge outwards, a pair of wedges with the thin edge inwards are then driven between the two first, then the fifth is inserted between the latter and driven up with a heavy hammer. This exerts a great pressure in the bore-hole upon the coal or stone (see figs. 96, 97).

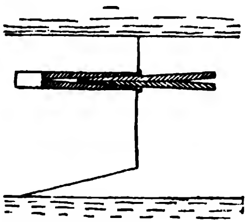


Fig. 96.—Patent Multiple Wedge when beginning work

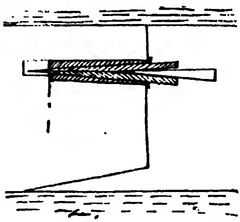


Fig. 97.—Patent Multiple Wedge with the fifth wedge inserted

Hydraulic Wedges.—These are expanded by water-pressure from a hydraulic pump or an accumulator. There are various makes, and for each some special advantage is claimed. Their chief disadvantages are: they

are slow in action, heavy to handle and convey about the mine, liable to derangements, and costly.

There are many other inventions of mechanical wedges which produce a rending force of less or more power, but very few of them are adapted for bringing down a mass of coal easily and quickly, and none of them have come into general use.

Lime Cartridges.—Lime increases materially in volume during slaking, and has been regarded as a source of power that might be utilized for bringing down coal, as a substitute for explosives. Specially prepared compressed lime cartridges are sometimes used. The substance is nearly pure carbonate of lime, calcined and ground to a fine powder, and then compressed into cartridges $4\frac{1}{4}$ inches long by $2\frac{1}{2}$ inches diameter, with a groove $\frac{1}{2}$ inch diameter along each. The method of using them is as follows: After the bore-hole is drilled, an iron tube, $\frac{1}{2}$ inch diameter and perforated, is put in. Then the required number of cartridges are pushed to the end of the hole, and stemmed in the ordinary way. Water is then forced along the perforated tube and escapes into the lime, which becomes saturated. After an interval steam is developed and the lime expanded, producing great pressure, which may finally force down the coal.

CHAPTER XI

THE DIFFERENT METHODS OF WORKING COAL

The shafts having been sunk to a seam or seams of coal and fitted up for drawing or winding coal to the surface, and the preliminary work of forming the shaft pillar, shaft sidings, and main roads having been carried out, the working of the coal must now commence, and

the method to be adopted must be well considered before proceeding.

Before deciding the question of method it is necessary to become acquainted with the nature of the seam, its roof and thill, and all other local conditions and circumstances that may affect the working of the seam. No method of working can be laid down to hold good in every case, or which will admit of general application, and the one that is found to give the greatest safety with economy should be adopted.

The chief methods of working coal, which, however, are subject to numerous modifications, are enumerated below, and shall be described in the order given.

Bord and Pillar; also known in different districts where it is adopted by the terms Bord and Wall, Post and Stall, Pillar and Stall, and Stoop and Room.

Longwall.

Single and Double Stall.

Square Work.

BORD AND PILLAR METHOD OF WORKING COAL

This system has been extensively adopted in North of England collieries. It comprises two operations, one known as the *whole working*, and the other as the *broken working*.

Whole Workings.—The whole working is carried on first, and consists in driving in the solid coal one set of excavations called *bords*, and another set at right angles to these called *walls*, *headings*, or *headways*. These excavations form blocks or pillars of coal which are more or less rectangular in shape (see fig. 98). The bords are usually driven across the cleat of the coal, and are the places that produce the chief output of coal. The headways, which are driven in the line of the cleat or "on the end", are driven for ventilative purposes chiefly. The bords are usually made "wide", as it is

termed, varying from 4 to 7 yards, according to circumstances. Sometimes they are made " narrow ", or only 2 or 3 yards in width, for special purposes. The headways are usually made narrow, except in some places where the cleat of the coal is very indistinct, in which cases they are made of the same width as the bords.

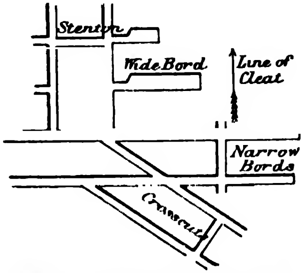


Fig. 98.—Excavations or Working Places by Bord and Pillar Methods

Another set of excavations sometimes driven in seams for particular purposes are called *crosscuts*. These are made at an angle to the bords and headways, or more or less diagonally across the pillars.

Narrow excavations and crosscuts are usually more expensive than wide places, because the coal is much more difficult to get or hew, and, in consequence, men are paid yard price or yardage for the distance they drive, over and above the usual price for getting coal.

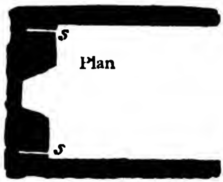


Fig. 99.—Method of Working a wide Bord when Blasting is resorted to

s s, Shot-holes.

Each bord or headways is usually driven forward by one coal hewer or getter in each shift, who, in many instances, does the hewing, blasting, and filling of the coal into the trams or tubs. The method of getting coal in a wide bord is first to *kirve* or under-cut it at the bottom of the seam for a depth of 3 or 4 feet, and from side to side of the bord. When the coal has been under-cut, it sometimes can be got down by the pick in large pieces without much labour; in other cases wedging or blasting must be resorted to. When blasting is required, a bore-hole is drilled in a suitable place near the roof, generally about the middle of the

bord, and the charge fired. This unkeys, or sumps. the face of the bord by taking out the centre piece of coal, and two lengths are left, one on each side of the bord, as shown in fig. 99. These lengths are next blasted, the position of the shot-holes being shown in the figures. For these side shots the charge of explosive does not require to be so large as for the middle or sumping shot.

Sometimes, in addition to the coal being under-cut along the bottom, it is cut or nicked in at one or both sides of the bord between roof and thill, as shown in fig. 100. This cutting unkeys the face, and two shots may be sufficient to do the work of three in the former case. This, of course, increases very much the work of the hewer or getter, as he has so much more cutting to do, which is the miner's most laborious work. In headways and other narrow excavations side-cutting on one or both sides is generally done in addition to under-cutting. When a soft band of stone occurs in any part of the seam, it may be easier and cheaper to kirve it out instead of kirving along the bottom of the seam.

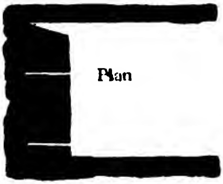


Fig. 100. — Method of Working when one side is Nicked

The coal produced by kirving and side-cutting is usually very small, and in some cases it is thrown back and left in the mine, only the "round" coal being sent out.

Size of Coal Pillars.—When it has been decided to adopt the bord and pillar method of working a seam of coal, a question of very great importance at once arises, viz. the dimensions of the pillars or blocks of coal to be formed by the bords and headways in the whole working. To determine this, it is necessary to take into consideration (1) the depth of the seam from the surface, because, with the depth, the pressure of the

superincumbent strata which the pillars will be required to support increases. Therefore, when the depth increases, the dimensions of the pillars must be enlarged in order to give them sufficient strength to support the roof, and prevent heaving or lifting of the floor.

(2) It is also necessary to take into account the nature of the coal. With a soft, easily-broken coal, it is evident that the pillars will require to be larger than with a hard coal, other things being equal.

(3) Then there is also the nature and strength of the stone forming the roof and thill, particularly the latter, to note. With a soft, yielding thill, the superficial area of the pillars should be large, so as to distribute the weight of the superincumbent strata over a large surface. The inclination of the seam is another matter that ought to be considered, a larger pillar being required where the dip of the seam is considerable than where it is slight.

It should be remembered that no hard-and-fast rules can be laid down for ascertaining the requisite size of pillars to support the roof that will apply to all seams, owing to the varying conditions under which coal occurs. Experience and knowledge of the various peculiarities of coal-seams, and a thorough study of local conditions, are needed in determining questions of this kind.

Pillars were formerly made much less in dimensions than they are made now; for instance, at a depth of 100 fathoms the usual size was 22 yards long by 10 yards wide, but at the present time very few pillars at that depth are made of less size than 30 yards long by 14 yards, and in some cases they are as large as 50 yards by 30 yards. In some collieries in the North of England, where seams are being worked at depths varying from 200 fathoms to 300 fathoms, the pillars are made as large as 60 yards square.

Much expense and trouble would often be avoided if pillars were always regarded as blocks of coal to be subsequently removed, and not merely as supports for the roof for the time being. It is better to err on the side

EXAMPLES OF THE DIMENSIONS OF PILLARS LEFT BY THE FIRST OR WHOLE WORKINGS AT VARIOUS DEPTHS

Name of Colliery.	Seam.			Pillars' Dimensions.	Width of working places.		Percentage of Coal.	
	Name.	Thickness.	Depth.		Bords.	Headings.	Taken.	Left.
		feet.	feet.	yards.	yards.	yards.		
Houghton ..	Hutton	4	1200	20 × 20	4	2	25	75
" ..	Main Coal	7	570	100 × 80	2	2	5	95
Haswell ..	Hutton	5	1200	30 × 20	5	2	25	75
Ryhope ..	Maudlin	7	1500	40 × 30	4	4	20	80
South Hetton	Hutton	5	1060	33 × 14	4	4	30	70
Townley ..	Brockwell	3	504	30 × 12	5	2	27	73
Allanshaw ..	..	7	702	30 × 20	4	3	24	76
Wearmouth ..	Maudlin	7	1620	40 × 40	4	3	16	84
Silksworth ..	Maudlin	5	1620	50 × 50	4	2½	12	88
Murton ..	Hutton	4½	1500	44 × 44	4	3	14	86
Garesfield ..	Brockwell	2	210	24 × 24	5	3	25	75
Clifton Hall ..	..	5½	1620	30 × 30	2½	2½	14	86

of strength by making very large pillars, than run the risk of disastrous consequences to the mine by having pillars with a very little margin on the side of strength.

Creep.—The danger in having small pillars is that they are liable to bring on *creep*. This is a heaving or lifting up of the thill of the seam in the excavated places, owing to the pillars being of insufficient dimensions to support the pressure of the superincumbent strata. This is the primary cause of creep, and usually it first shows itself by a slight heaving on the roads, sufficient to break the timber and alter the level of the tram- or wagon-ways. The pillars may appear to be strong enough for a time until the workings have become extended and the limit is reached at which the pressure is too great for the pillars to bear, then the downward movement of the upper strata commences and forces the pillars into the floor, which rises up in the excavations until they become choked up and completely filled.

Another cause of creep is a soft, yielding thill or floor. This may give rise to creep even when the pillars are large enough for ordinary circumstances, and the danger may be further increased by the presence of water, which

in some cases causes the floor to swell, and so renders it more liable to heave.

Faults sometimes induce creep; to avoid it, pillars ought to be made larger than usual in their neighbourhood. The strata at faults are frequently much broken up, and the conditions are favourable for disturbances.

Pillars should in all cases be made of such dimensions as to render it practically impossible for creep to ensue, and in working the coal great care should be exercised.

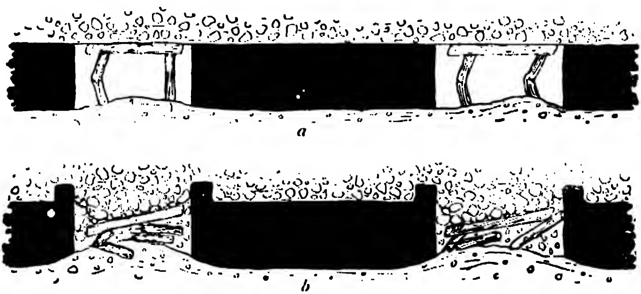


Fig. 101.—"Creep" in a Coal-seam

a, Shows creep in its initial stage. *b*, Shows creep in its final stage.

because, apart from the causes of creep already mentioned, it may be produced by an injudicious method of working the broken. Fig. 101 shows creep in its initial stages, and also in its final stage, when it has closed up the excavation and become settled.

Thrust.—When the bed forming the thill of a coal-seam is dry, hard, and unyielding, and pillars are left of insufficient dimensions, a disturbance variously known as *thrust*, *crush*, and *sit* may occur. The thill being hard does not heave in the excavations, but the pillars of coal being weak, are cracked, crushed, and ground between the descending roof and the unyielding thill.

The Laying-out of Workings.—The direction of the main roads, such as wagon-ways and engine-planes,

in the seam, is determined to a great extent by the position of the royalty or royalties to be worked. The different districts, as a range of working-places is termed, are usually arranged or set off from the main roads, so that the bords may be driven across or at right angles to the cleat of the coal, and the headways in the direction of the cleat.

In many cases the workings are made to follow each other continuously over the entire area to be worked. Another plan is to arrange for barriers or solid ribs of coal being left at certain distances apart, so as to divide the workings of the mine into a series of separate districts or panels, as it is termed. This arrangement was first introduced by Mr. Buddle, and although possessed of many advantages, it is not so extensively adopted as might be expected. Fig. 102 is a plan of a seam worked by the bord and pillar method laid out in the panel system, and shows the barriers and districts into which the workings are arranged. The advantages claimed for this arrangement are: each district being isolated and enclosed by barriers, if creep occurs in one district the barriers may prevent its extension to any other; the ventilation of all the workings is simplified and improved, because each panel has its own intake and return; the effects of an explosion which might occur in one panel may not extend beyond the one in which it takes place; also the safety of the mine generally is increased.

Broken Working.—The second operation in the bord and pillar method consists in the working away or removal of the pillars which were formed by the whole workings, and is known to miners as the "broken working". Nearly all the coal in the seam is by this means secured, the amount lost depending upon the method in use of removing the pillars, the local conditions, and the care exercised.

In the early history of coal mining, it was the universal practice to abandon the pillars in the mine when the boundaries were reached in the whole workings, the

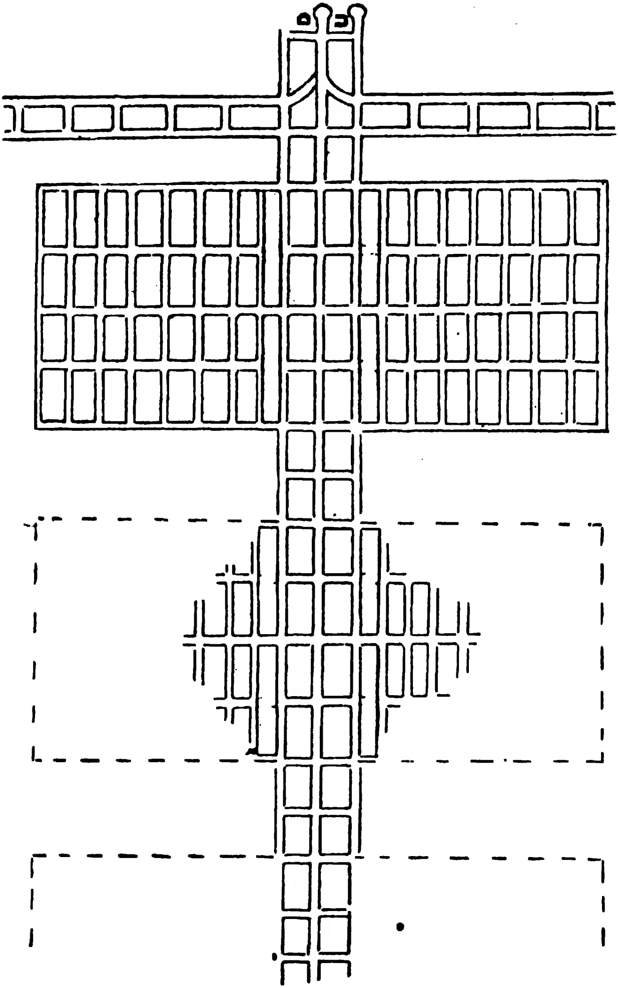


Fig. 102.—Bord and Pillar System arranged in Panels

D, Downcast shaft. U, Upcast shaft.

method of broken working not having been attempted. In this way much valuable coal was lost, although the pillars were made of the smallest possible dimensions that would maintain the stability of the strata until the boundaries were reached. Many of the workings of these old mines have since been entered and the best of the pillars removed. Since the removal of pillars was found to be possible, the practice has been universally adopted in the bord and pillar method.

The work of removing the pillars is sometimes not commenced until the boundaries are reached in the whole; but more often the process begins at a safe distance from the shafts, and follows up the advance of the face of the whole workings. Three or more pillars of distance must be kept between the whole workings and the gob or goaf, the latter being the area from which the coal has been entirely removed by the broken working and the roof allowed to fall in (see fig. 103).

It is generally admitted that the "following up" system of the broken is better than that of leaving the pillars untouched until the boundaries are reached and the whole workings completed. Pillars left standing for a long time undergo deterioration, become crushed, the roof of the excavations frequently breaks down, and thus very much increases the cost of subsequent working. By following up the whole at a convenient distance all this is obviated; the pillars, having to stand for a short time only, are not crushed or deteriorated, and the roof in the bords and headways is left upstanding. Other important advantages in connection with the following-up system are: the ventilation of the mine is more satisfactorily accomplished, as the air has not so far to travel or so many air-ways to ventilate; the cost of repairs to air-ways is not so much, as there are not so many to maintain; the men and working places are more concentrated; and the rails used for working the whole mine are in many cases still left on the roads for use when the broken follows up.

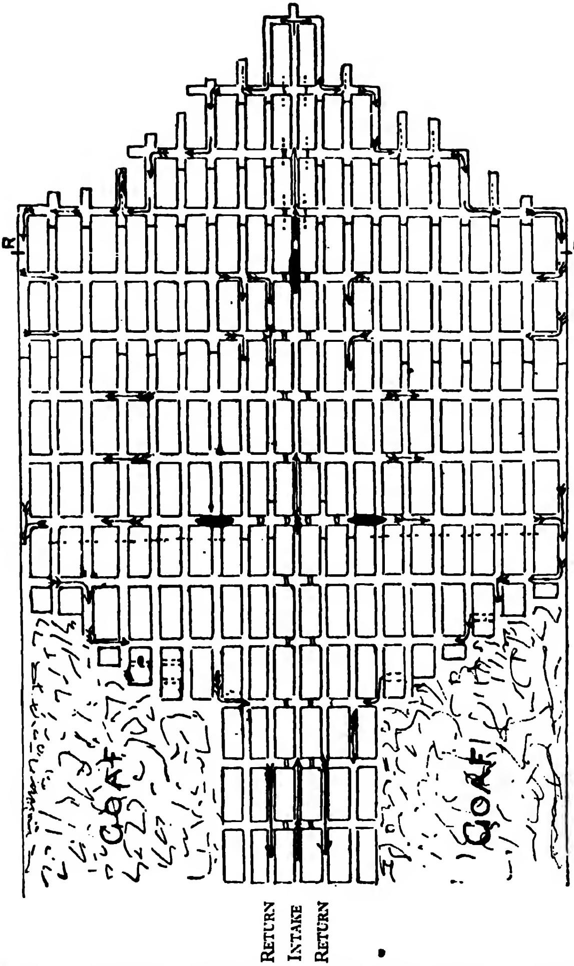


Fig. 103.—Illustrating method of following up "Whole" with "Broken" in Bord and Pillar system of working coal, and the Ventilation of the workings

REFERENCE TO DETAILS. Air currents Doors, D. Regulators, R.
STOPPINGS. Canvas Wood Brick or stone
Flats or sidings in which the coals are brought by Putters

Of course, as every mine has its own peculiar local conditions, the system best adapted for those conditions is applied; and, in consequence, we see both the following-up system, and that of waiting until the boundary is reached before starting the broken, practised at different mines or even in the same mine.

Methods of Removing the Pillars of Coal.—The following are some of the methods adopted for removing pillars. No rule can be laid down for the selection of any one method as being the cheapest and most satisfactory; it often requires a practical test to ascertain which is most suitable for the circumstances of each case.

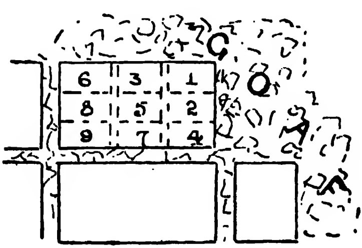


Fig. 104.—Method of removing or working in Lifts a Pillar of Coal

1. The pillars are removed in lifts or slices right and left from each headways. Each lift is driven parallel to the bords for half the length of the pillar, the width of the lifts varying from 4 to 7 yards, according to the nature of the roof. Each lift has the goaf on one side, and as soon as it is driven the required distance, the timber which has been put in to support the roof is drawn out and the latter allowed to fall, forming more goaf. This plan is objectionable; much timber is lost, and the ventilation at the face is defective.

2. The pillars are removed in lifts, which are always driven in one direction from each headways. The pillar is first divided into three or more equal portions by holing walls, which are driven from the old bord over to the goaf. At the end and side of the pillar nearest the goaf the lifts are commenced, and are taken off or driven in rotation back to the headways. This may be better understood on referring to fig. 104, in which the

lifts are numbered and are removed in the order given. No. 9 is the most dangerous to remove, and requires to be strongly timbered with props and chocks and carefully watched. This is a safe method of removing pillars where the stone is " heavy ", but the straight or narrow work is costly.

3. Another plan is to drive a *jenkin* down the middle of the pillar in a bordways direction, and when it reaches the far end of the pillar, the two sides are worked back simultaneously. Two or three rows of props and chocks are used behind the face, and the back row is drawn out as the face works back, and a fresh row put in. This method is liable to cause lifting of the floor, especially in the jenkin, causing much trouble and expense. Sometimes the coal is worked from the jenkin in right- and left-hand headways lifts (see fig. 105).

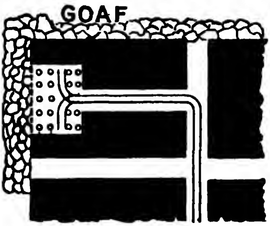


Fig. 105

4. A plan sometimes found to work satisfactorily (and one that is cheaper, as there is no narrow work required) is to take off the pillars by a longwall method. The tramway is laid down the old bords from the headways, and along the end of the pillars, which form the face to be worked, back towards the headways. The face is timbered as in a longwall. This method, however, requires height of seam to admit the tubs along the face to be loaded.

Timbering Whole and Broken Workings.—The working of coal by the bord and pillar method being carried out in two operations, namely whole and broken, the methods of timbering followed in each case must be considered. The roof stone in the excavations formed by the whole workings may in some cases require very little support, as the pillars are left of such dimensions as to obviate any movement of the overlying strata. But

it is obvious that when the removal of the pillars commences, the roof losing their support breaks down and causes movement for some distance from the goaf. Consequently, in removing the pillars, it is necessary to timber very carefully with stout props and chocks, even if the roof has been so strong as to require very little timber in the whole.

Whole Workings.—The bords and headways are timbered at once as they are being driven, and the timbering is kept close up to the face in regular order. The ordinary crowntrees, about six feet in length, are placed transversely across the excavation, with a prop under each end. The tramway is usually laid under this timber; and between this and the side of the coal, props with a broad head-piece or cap-piece are fixed (see fig. 106).

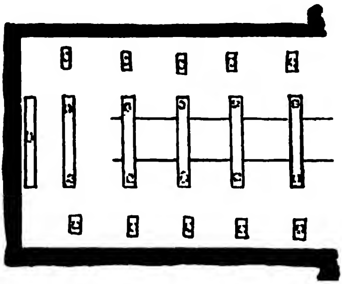


Fig. 106.—Illustration of Timbering a wide Bord

The actual amount of timber necessary in each place depends upon the nature of the stone, some roofs being so hard and strong as to require little or no support; in others the stone is so "short" and tender as to break down between the timbers, even when they are placed quite close together.

The state of the roof is ascertained by rapping or *jowling* it smartly with an instrument, such as a hammer or pick, and by careful examination. When the rapping of the roof emits a clear ringing sound it is said to sound "good", that is, there is no immediate danger of a fall; when the sound is dull and hollow, the stone is heavy and liable to fall, and should be timbered. A clear ringing sound, however, is not always to be relied upon, because the roof may contain slips, joints, and cracks,

and fall away at these. It is necessary, therefore, to look carefully for these in addition to rapping, and, where they are detected, to put in timber of suitable size and strength. Some roofs contain a number of *pots* or *cauldron bottoms*, cone-shaped masses of stone, with the narrow part upwards and the sides usually glassy and slippery. These usually sound good owing to their thickness, but they are very liable to drop out unexpectedly. When seen they should either be strongly timbered, or, better still, taken down.

Systematic Timbering.—The Mines Act specifies that, in order to minimize accidents from falls, the roof under which coal-getting or tub-filling is carried on shall be systematically and adequately supported, and the props or chocks shall be set at such regular intervals and in such a manner as may be specified at each mine.

On main roadways the roof and sides shall be systematically and adequately supported at such regular intervals and in such manner as may be specified.

(1) *Propping of Roof.*—The maximum intervals on roadways between the supports, and at the face, must be specified as follows:

(*a*) Between each row of props in each direction.
(*b*) Between the front row of props and the face.
(*c*) Between the holing props or sprags.
(*d*) Between chocks.

These must be enforced whether the roof is considered " good " or " bad ". Where necessary for safety, supports should be set nearer than specified.

(2) *Supply of Timber.*—The officials of the mine are responsible for having a sufficient supply of suitable timber for supports within ten yards of every working place.

(3) *Drawing Timber.*—Whenever timber has to be withdrawn from the waste, goaf, or insecure roof, a safety contrivance shall be used, such as a " ringer and chain ".

Broken Workings.—The roof of the broken workings is generally in a more unsafe condition than in the *whole*, owing to the entire removal of the coal and the downward movement of the overlying strata. All excavations where persons are required to travel in a broken district should be timbered, and all lifts as they are being worked should be carefully timbered close up to the face in a systematic manner. A large number of props with crowntrees and headtrees are generally used, and they should be so placed as to cover all faulty places and do the most good in sustaining the roof. When the roof is strong and falls in huge blocks or frames, chocks or cogs must be used in the lifts. These are pieces of hard wood about 22 or 24 inches long by 4 or 5 inches square section, built up in squares by placing two and two crosswise, as shown in fig 107 These chocks are capable of resisting great pressure of roof.

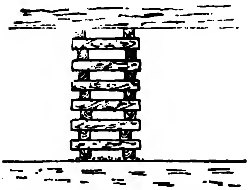


Fig. 107.—Wood Chocks to support Roof

The withdrawing of the timber out of lifts which are finished requires great care, and is generally entrusted to steady, experienced workmen. The "drawing" of the timber is commenced at the edge of the goaf, and it is gradually taken out back to the next pillar or portion of a pillar of coal. The uninjured timber and chocks may be used again; usually, however, owing to the weight of the roof or the lifting of the floor, much of it becomes bent and broken and cannot be used again. No timber should be left standing in the goaf, except pieces that it would be absolutely dangerous to get or knock out; to leave any standing may cause the weight

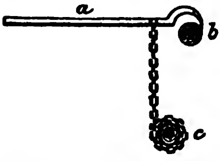


Fig. 108.—Ringer and Chain

a, Lever. *b*, Fixed prop used as fulcrum. *c*, Prop with chain around to be withdrawn.

to be thrown upon the next lift. The appliances for withdrawing timber are: (*a*) deputy's mall, (*b*) the ringer and chain (see fig. 108), (*c*) Sylvester's patent prop withdrawer, and (*d*) the "Hardy" pit prop puller (see fig. 109).

Fig. 109.—"Hardy" Pit Prop Puller. The prop marked X X X is to be withdrawn

Ventilation.—The ventilation of the workings of a seam worked by the bord and pillar method, especially in an extensive mine where firedamp is given off, is a subject which requires much thought and attention. It is necessary to carry out the provisions of the Act with regard to supplying an adequate quantity of air, to keep the roadways and working places of the mine in a safe

condition for working and travelling in, and to dilute and carry away all noxious gases given off.

This subject is more fully explained farther on, but it may be stated that the air currents are conducted along the principal roads into the various working districts. The current for each district is usually carried up the middle and leading working place to the face, and made to split into two portions, one going to the right and the other to the left. Each portion is conducted along the face headways, and up to the face of each bord. It is conducted up to the face of a bord or other excavation, by dividing the latter into two parts, by means of a vertical partition carried along to within a few feet of the face (see fig. 110). This vertical partition reaches from floor to roof, and consists of brattice cloth or canvas, or in some cases of thin wood deals, which may be " pointed " or plastered at the joints with lime, to make it as air-tight as possible. The air travels up one side of the partition and passes around the face of the excavation, and returns down the other side. After ventilating a bord, it travels along the headways to the next bord, up which it passes to the face, returning again to the headways, and so on until it reaches the last bord in the district, whence it passes into the " return ", along which it travels to the upcast shaft.

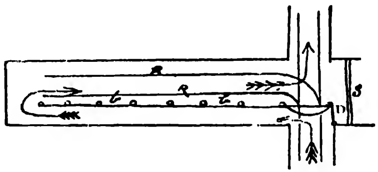


Fig. 110.—Illustrating method of conducting Air to the face of a Bord or other Excavation. The arrows indicate direction of air current

b b, Brattice fixed to props. D, Brattice door. *s*, Stopping. R R, Line of rails.

After ventilating a *whole* district the same air is sometimes conducted to a *broken* district to be used again, but sometimes fresh air is used in both. It is unsafe to carry air from a broken to a whole district, and from places where *safety lamps* are used to where *naked lights* are allowed to be used.

CHAPTER XII

THE LONGWALL METHOD OF WORKING COAL

In this method of working coal the chief principle is the extraction of the whole of the seam in one operation, and allowing the superincumbent strata to settle in the gob or goaf behind. It is adopted more or less in all the coal-fields of the United Kingdom, but chiefly in those of Yorkshire, Derbyshire, and Lancashire.

Longwall may be divided into two kinds:

I. Longwall advancing or working away from the shafts.

II. Longwall retreating, or working back from the boundaries towards the shafts.

I. LONGWALL ADVANCING

In this method, the total extraction of the seam commences from the shaft pillar, which must be left intact to maintain the stability of the shafts. The coal is removed in a long length or face, which may be in one straight line or in a series of steps, as shown in fig. 111. The arrangement of the face in a straight line or in steps depends upon circumstances; some seams are found to work better, that is, the coal is easier and cheaper to get by having the face in a straight line, and others by having it in steps. The line of the face is to some extent determined by the direction of the cleat of the coal; the coal in most cases works better " face on ", that is, when advancing across the line of the cleat. Sometimes in order to get round coal, the face is carried forward " end on " or in the line of the cleat, and sometimes better results are obtained in a crosscut direction.

The coal is under-cut, holed, or kirved along the face by hewers or getters, or by machine coal-cutters. This

work is usually done in the bottom part of the seam, or if there be a bottom layer of soft stone or band, the under-cutting may be done in it. The depth to which the coal is under-cut by hewers varies from 3 to 6 feet, and is in the form of a wedge, as shown in fig. 112. During the kirving or under-cutting, the men are pro-

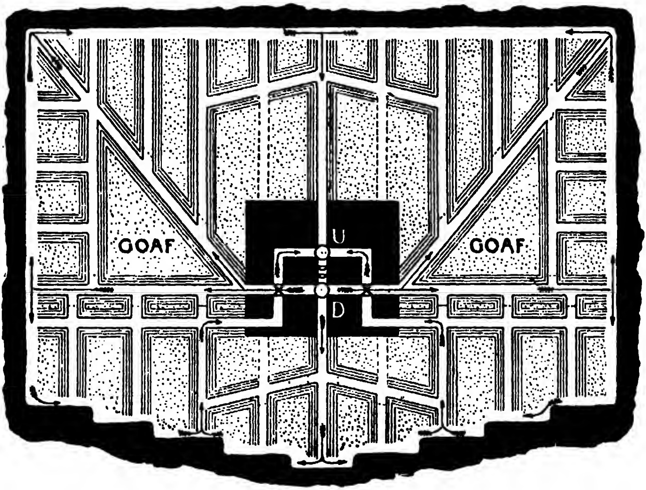


Fig. 111.—Illustrating Longwall advancing with Straight and Stepped Faces, Shaft Pillar, and Ventilation

D, Downcast shaft; U, Upcast shaft; Air currents; Wood doors: Air crossing; Canvas doors. Gateways; lines indicate the packwalls of stone.

tected from a fall of the coal by short props, termed sprags, set at a distance of not more than 6 feet apart along the face under the coal as the under-cutting proceeds (see fig. 113). The form of sprag shown in right-hand sketch is known as the "Cockermeg", and is very useful in thick seams.

As the face advances, roads must be formed and kept

open through the gob or goaf to communicate with the shafts. This is done by building walls of stone, termed packs or packwalls, on either side of the road from the floor to the roof. The stone for this purpose is obtained from the roof, which is usually taken down in the road-

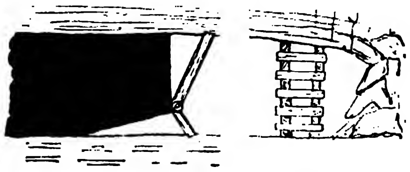


Fig. 112.—Under-cutting, Spragging, and Timbering of Longwall Face

ways to make height for tubs to travel, or from the floor, which is in some cases taken up to make height, or from the fallen roof in the goaf. In some instances band or stone occurring in the seam is utilized for the purpose, and in others stone may be brought from another part of the mine or from the surface. The

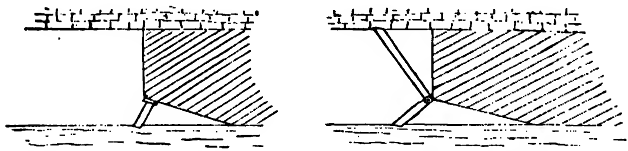


Fig. 113.—Two Methods of Spragging Coal which has been Under-cut or Kirved

roads formed by these packwalls are termed *gateroads* or *gateways*. As the face advances, and the area of the goaf, from which all coal has been extracted, increases, the upper strata begin to subside and put weight upon the packwalls, which are pressed tight and forced down into the floor. The necessity for making the packwalls solid and substantial is then seen. Their width varies according to circumstances, but in no case should they

be made less than 6 feet, and built solid from side to side. In some cases thin walls of stone are built up on the gateway side and goaf side, and the space between filled with any kind of loose stone thrown in, but this plan is not satisfactory.

Fig. 114 is a section of a gateway with the packwalls on each side newly built, and before any weight of the upper strata has settled upon them. Fig. 115 is a section of a gateway showing the change that takes place when the roof settles down through the extraction of the coal. The packwalls become squeezed very tight, and the height of the gateway becomes so much reduced as to necessitate ripping down some of the roof to make sufficient height for haulage requirements. The gateways in time resemble stone drifts, owing to the squeezing of the packwalls and the amount of roof stone that has to be ripped down.

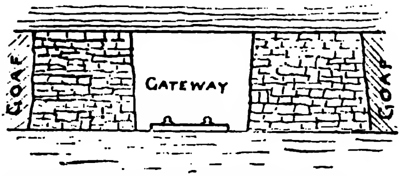


Fig. 114.—Gateway with Packwalls newly built

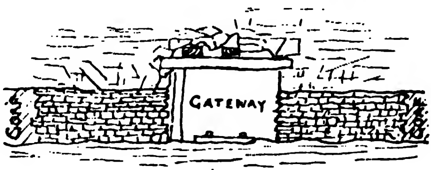


Fig. 115.—Gateway with Roof ripped down to make height, and showing the effect of Weight on the Packwalls

The width of the gateways, or the distance between the two packwalls, is determined by the requirements of the mine, but is seldom less than 6 feet. The distance the gateways are apart depends to a large extent upon

the height of the seam. If the latter is of such a height as to allow the passage of the tubs or trams along the face to be filled without requiring the removal of any roof or thill, then the gateways may be placed from 20 yards to 50 yards apart, but when the seam is of less height than the height of the tubs, so that they cannot pass along the face, the gateways must be made nearer together, say from 10 yards to 15 yards. For as the tubs stand to be filled at the end of the gateway near the face, and the coals are cast or shovelled along to them, it is obvious that unless the gateways are placed near together in a thin seam, the work of casting or shovelling the coals along the face becomes inconvenient and costly. Accordingly as the face advances, the gateways are advanced to keep within a few feet of it (see fig. 116), in order to have the tubs near at hand for easy filling, and also to have the roof supported by the packwalls.

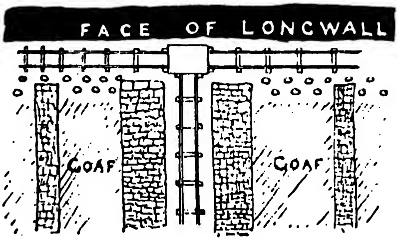


Fig. 116.—Plan showing Packwalls and Timbering at Face of Longwall, and Tramroad laid along the Gateway and along line of Face

When a number of gateways have advanced a considerable distance, it becomes a difficult and expensive matter to maintain them, and inconvenient for the haulage of the tubs from the face of each gateway. One main gateway is then maintained, and a "cross-gateway" is formed across the other gateways and at right angles, or nearly so, to them. The cross-gateways enable the old gateways to be dispensed with, whereby the distance of haulage and the cost of maintenance is reduced. The ordinary gateways are carried forward from them as shown in fig. 111.

Complete stowage of the goaf in longwall working ensures success. If the space previously occupied by the coal can be filled up regularly and completely by any rubbish there may be in the mine, the roof will settle down evenly over the district, and cause less damage to the gateways, and, consequently, the maintenance of the latter for repairs, timber, &c., will be less costly.

Incomplete stowage allows the roof to fall and break up between the gateways, putting side pressure upon the packwalls, and otherwise causing trouble and adding to the dangers of the mine. Where possible, all refuse, such as dirt bands, falls of stone, and spare packing, should be thrown into the goaf to fill it as much as possible, but on no account any material that may be liable to spontaneous combustion.

Timbering.—The regulations for systematic timbering quoted on pages 139 and 140 apply to longwall workings as well as to bord and pillar. Along the face of a longwall it is necessary to put in temporary supports systematically, in order to protect the men when working from falls of the roof. Two or three rows of props are set along the face at regular intervals from 2 to 4 feet apart. The props should not be opposite to each other in the three rows, but alternately, and in this way a larger surface of roof is supported. The timber should be kept close up to the working face in a regular and systematic manner, so as to allow the weight to spend itself in the goaf, and prevent it coming upon the face and timber. When the stone is very "heavy" or "weighting" very much, chocks are used in addition to the ordinary props and headtrees or cap-pieces. As the face advances, and a fresh row of props or chocks is put in, the last row nearest the goaf is knocked out, and may be used again. In this manner the timber is regularly shifted forward and the roof allowed to fall or settle down in the goaf between the gateways.

Great care should be taken to avoid leaving any

timber in the goaf, because, in addition to being wasteful, it prevents the roof breaking and settling down regularly and uniformly, and may be the means of throwing excessive weight upon the face timber and the coal.

The gateways also usually require timber support between the packwalls. The ordinary crowntrees or half-balks resting upon two props, or supported by the packwalls, are generally used. As the roof settles down upon the packwalls it becomes much broken in the gateways, and necessitates frequent renewal of the timber, until the goaf becomes pressed tight and all movement ceases.

II. LONGWALL RETREATING

This method consists of driving winning places, levels, haulage roads, and air-ways to the boundaries, and when the latter are reached to extract the coal by the longwall face and work back in the direction of the shafts. The roads communicating with the shaft being already made in the coal, as the face retreats the roof is allowed to fall completely, and no roads whatever are maintained through the goafs. The face is timbered in a similar way to that explained in the previous system, and the timber is all withdrawn as the face recedes. It will thus be seen that the work is very much more simple than when gateways are made through the goafs, as in the previous system.

Comparison of Longwall Advancing and Longwall Retreating.—As to which of the two systems is preferable, it is difficult to say. Speaking generally, longwall advancing is more frequently adopted than longwall retreating. The chief objection to the latter method is the prime cost of driving the requisite number of haulage- and air-roads to the limits of the royalty, and the delay thus occasioned in obtaining the expected output of coal. There is, however, to be considered the benefit derived from the amount of knowledge of the coal-field that is acquired; of the nature, quality,

and thickness of the coal to be worked over the whole of the royalty; of the nature of roof and thill; and the direction and amount of throw of faults. Another advantage of the retreating system is, that if the seam contains firedamp, it will to a large extent be tapped, and thus the quantity to be subsequently given off by the longwall face reduced.

The longwall advancing method is more applicable to the working of thin seams than of thick ones, especially where dirt bands or other stone is available for the packwalls. The first construction and subsequent maintenance of the gateways is generally costly, especially if the seam is thick and the stone for packwalls difficult to procure.

The retreating system is better adapted than the advancing for the working of thick clean seams, as it requires no packwalls or gateways. It is also advantageous in a seam containing many faults or other interruptions of the regular continuity of the seam—that is, if other conditions favour the longwall method in preference to some other, say the bord and pillar method. Seams having very soft roof stone, and lying at a high angle of dip, are better worked on the retreating than the advancing system.

Seams liable to spontaneous combustion resulting in "gob-fires" should, if possible, be worked on the retreating system, because the gob being always behind, if a fire breaks out, the longwall working face is always receding farther from it; whereas with the advancing system the gob-fires occur between the shafts and the working face.

Ventilation of Longwall Workings.—The distribution of the air currents in a mine worked on the longwall method is carried out in a very simple and easy manner, and the simplicity of the ventilation is a great advantage in this method of working coal.

The air is taken straight along the main gateways to the face, where it is usually made to split into two portions, one going to the right and the other to the left.

Travelling straight along the face, the air currents sweep the gas and other impurities along into the return airways. Very few doors, stoppings, and brattices are required to get the air up to the face.

The chief drawback is the usual stagnant condition of the air in the gateways other than the main ones. The air is generally conducted up the centre gateway, whence it travels straight along the face without traversing the other gateways, which are only ventilated by leakages of air. For more information on this subject, see the chapter on Ventilation.

Distribution of Labour at the face of Longwall.—In the bord and pillar method of working coal, each man or coal-hewer usually carries out all the three operations of work connected with the getting of the coal. These are *holing* or *under-cutting*, *getting* or *breaking down*, and *filling* or *loading-into-tubs*. He is usually paid a certain price per score of tubs or per ton loaded and sent to the surface, and in this price is included his remuneration for under-cutting and breaking down. This arrangement is sometimes carried out in longwall working, but does not give satisfaction. A better arrangement, and one very much adopted, is to divide the three operations or classes of labour among three sets of workmen. The first set are the most skilled workmen, who do the under-cutting or holing, and are usually termed *holers*. They under-cut along the face, and are paid a certain rate per yard measured along the face, the depth of the under-cut being arranged and fixed. The holers are followed by the second set, who break down the face, or, in other words, do the actual getting of the coal, and are termed *getters*. They are generally paid a certain wage per day. These men are followed by the third set, who fill or load the tubs or trams with the coals lying at the face, and are termed *fillers*. They are paid upon the quantity loaded. The work of packing and timbering may be done by another set of men.

Another arrangement is for three or more men to share the work and the money resulting from their labour. They agree to work a certain length of coal face amongst them, doing all the work connected with holing, getting, and filling.

Sometimes an agreement is made with one man to be responsible for the working, timbering, and packing of a certain length of face, say the length of one gateway, and to find and pay all the labour required to do the work.

CHAPTER XIII

COMPARISON OF BORD AND PILLAR AND LONGWALL

Varying Conditions.—The conditions under which coal occurs are so varied and complicated, that it would be most unwise to assert that this or that system of working will give the best results in every case. Seams differ in thickness, quality, and kind; are found at various depths and at varying angles of inclination; and have different kinds of stone forming roof and thill. Some seams are better worked by the bord and pillar method, and others by the longwall, or by some modification of either of these methods. Each method possesses its own advantages, which render it applicable in certain cases. In every case, before determining the method to be adopted, all the circumstances should be carefully considered; and that method should be applied which is most applicable to the existing conditions so far as they can be seen, and which will produce the largest quantity of coal at a minimum cost and with maximum safety.

It should be noted that the longwall and bord and pillar methods are at present carried out in seams vary-

ing in depth, in section, in the nature and quality of coal, in nature of roof and thill, and in inclination of the strata, consequently an exact comparison cannot be made.

General Comparison.—The methods may be compared under the following heads: 1. Ventilation; 2. Haulage; 3. Produce; 4. Cost; 5. Safety; 6. Surface Conditions.

1. Ventilation.

Bord and Pillar.—In this method the air in a whole district is usually carried up the middle working place, and splits at the face right and left. It is not allowed to pass straight along the headways to the return, but is conducted to the face of each bord and working place by means of brattices. Thus it travels a considerable distance along a circuitous route in each district, and has a large rubbing surface, which, of course, means a large amount of friction, necessitating great ventilative power. A large number of stoppings, doors, and brattices are needed in order to direct the air into the various workings. Large volumes and blowers of gas can be isolated and much better dealt with than in longwall.

Longwall.—As already explained, the air is usually carried up the middle gateway to the face, where it splits, and travels right and left along the face of the longwall to the end of the district, and then goes direct into the return. This is a very simple arrangement, and needs few stoppings, doors, and brattices. The air has a short distance to travel, and consequently there is a short length of rubbing surface, producing a small amount of friction, and requiring little ventilative power. On the other hand, gas given off at any point is carried along the face to the return over the men. Blowers of gas appear to be of more frequent occurrence than under the bord and pillar system. The gateways are frequently ventilated only by leakages, as the main current passes straight along the face.

2. Haulage.

Bord and Pillar.—The conveyance of the coals from the face to the sidings or stations is long and circuitous, but usually the roads are easily maintained in good condition.

Longwall.—The coals are conveyed from the face along the main gateways and crossgates to the sidings, the roads usually being short and direct.

For the same output or quantity of coal, the haulage roads are much longer in bord and pillar than in longwall, as in the former the men are placed farther apart.

3. Produce.

Bord and Pillar.—In the whole working there is much narrow or straight work, and much niching or side-cutting which results in producing a large percentage of small coal. In the broken workings the coal is usually broken and crushed, and produces much small. A portion of the seam is frequently lost in the broken, owing to falls, and to the difficulty and danger of taking the coal clean off by the side of fallen excavations and the goaf.

Longwall.—The coal is all extracted in one operation, and little is lost or wasted. It is got in the best marketable condition, the percentage of round coal being high. The weight of the roof assists in breaking down the coal at the face, but does not rest upon it long enough to crush and break it, as the face is continually on the advance. Moreover, as longwall advances into the solid coal in one long face, very little narrow work and side-cutting is needed, consequently the conditions are favourable to the production of a maximum percentage of round coal.

4. Cost.

Bord and Pillar.—The price paid for hewing or getting the coal is usually higher in the whole mine than in the broken. In the latter, the coal is usually crushed and more easy to get than in the former, and a smaller price

is therefore paid. But taking the average price of the whole and broken, it will usually be more than that required in a longwall. Narrow work is required both in whole and broken, which adds materially to the cost of hewing. The cost of shift and stonework is a very variable quantity, depending upon the height of the seam and the nature of roof and thill.

Longwall.—The cost of hewing or getting the coal is usually small, as the pressure of the roof assists the working of the coal. There is little or no narrow work, and therefore no yard prices to pay. The work of holing or under-cutting, getting, and filling is usually divided among three sets of workers, and they become proficient in their respective classes of work, and therefore do it well and cheaply. The conditions are usually favourable for the application of machine coal-cutters. The cost of tramming, and also of materials such as rails, sleepers, timber, and brattice, is also less than in bord and pillar.

On the other hand, the cost of stonework in the blasting of the roof and the building of packwalls is usually very high; cases have occurred in which this cost has more than counterbalanced the gain from the lesser charges already enumerated, and the total cost has exceeded that of the bord and pillar.

5. **Safety.**—Falls from the roof are less likely to occur in longwall than in bord and pillar, as there is very much less roof exposed in the former than in the latter. In the bord and pillar method the excavations are timbered as they are being driven, and the timber is left standing until the places are no longer needed for travelling in. This may mean some weeks, whereas in longwall timber is put under a fresh roof each day and the back timber drawn out.

Some longwall gobs appear to be liable to spontaneous ignition, giving rise to gob-fires, perhaps owing to small coal being thrown back into the gobs. Also gas is more difficult to deal with, although better ventilation

may be secured owing to the short distances the air has to travel.

6. **Surface Conditions.**—The *bord and pillar* method may be worked under large bodies of water, or under upper seams containing water. It may also be worked under towns without damage to the buildings, provided the pillars are left in permanently. The longwall method is totally inapplicable to cases of this kind. Longwall, however, does less damage to the surface than bord and pillar when the broken is worked. In the former the face advances regularly, and the roof is let down evenly over a wide area. Thus the surface subsides gradually, uniformly, and without any of the numerous fissures and irregular depressions which are the result of working the broken in the bord and pillar method.

Other Points of Comparison.—Although both systems are worked under all kinds of roof, yet a roof which falls well, and is easily ripped down for height and packing, is more suitable for longwall. A very hard and framy roof, or a very soft, friable one, is unsuitable for longwall.

Bord and pillar is practised under all conditions of roof, although it is more suitable for seams having a hard roof. Seams of from 4 feet to 6 feet, free from thick interstratified bands of stone, at a moderate angle of inclination, are favourable for bord and pillar. Again, seams much intersected with faults, and those yielding much gas and water, are, generally speaking, better worked by the bord and pillar method.

Longwall is applicable to thin seams up to about 4 feet, or for very thick seams; also to seams interstratified with bands of stone, which may be used for packing, and to seams having any angle of inclination.

Summary.—*Bord and pillar* is suitable for working coal under difficult circumstances underground, and where the surface must remain undisturbed; under

certain circumstances, it compares favourably as regards cost with longwall.

Longwall gives a maximum yield of round coal, is simple to work and ventilate, and is, as a rule, less costly. Generally speaking, it is the better system of the two, is now very largely adopted, and is gradually superseding bord and pillar.

CHAPTER XIV

METHODS OF WORKING COAL—CONTINUED

Single Stall and Double Stall. Square Work Method for Thick Seams. The Working of Thin Seams and Highly-inclined Seams.

The **Single-road Stall** and the **Double-road Stall** methods of working coal are practised chiefly in South Wales, but have also been tried in other districts for the working of thin seams. Both methods admit of modification to suit the varying conditions of seams.

The Single-road Stall System.—The main roads from the shafts are usually driven as in the ordinary bord and pillar system, but sometimes they are made 8 yards wide, and the middle part is stowed or packed with stones to support the roof as the face advances, leaving a narrow road next the coal on each side, one forming the intake and the other the return. At intervals of 100 yards or upwards, headings are driven away at right angles to the main levels. These headings are also made 8 yards wide and are packed, a road being left on one side next the coal for the tubs and the other for the air. Out of these headings single-road stalls are driven, and ribs or pillars of coal are left between the stalls. The stalls are started about 6 feet wide, and are driven that width for a few yards, so as to leave more

coal near the heading; they are then increased in width to 8 or 12 yards. The actual width of the stalls, and of the pillars of coal left between them, depends upon the amount of stone yielded for packing purposes by the seam, and upon the depth from the surface and the nature of the roof and floor.

The stalls are packed down the middle, and a tram-road is left on one side, and an air-way on the other. A small opening is made in each rib, to connect the stalls so that the air current may pass through.

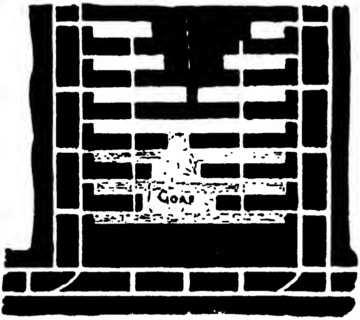


Fig. 117.—Single-stall Method of Working Coal, showing Stalls being driven and Ribs being worked back

The stalls are sometimes driven to within a few yards of the next heading, into which they are holed by a narrow place. Sometimes they are driven right and left from each heading, and hole into each other at half the distance between the headings. When each stall has reached its distance, half of the pillar on each side of it, or all the pillar on one side, is brought back or removed backwards to within a few yards of the heading. All timber is drawn out and the goaf behind allowed to fall (see fig. 117).

The Double-stall Method.—The mainways and headings are arranged similarly to those described in the single-stall method. The stalls are opened out from the headings by *two* narrow roads, 12 yards or upwards apart. These two roads are driven a short distance from the heading, and are then connected by a holing. This holing forms the face of the stall, which is then driven forward by two men, and in the space behind

a 6-yard packwall is built, leaving a road on each side in a line with the narrow openings. The air enters the stall by one of the openings, travels to the face, passes around, and returns along the other road back to the heading (see fig. 118).

Ribs of coal are left between the stalls about 12 yards wide. When the stalls are driven as far as they are required, the pillars are then worked off and the roof behind falls. Half of the rib on the right and half on the left is worked back by each stall. Small pillars or stooks are left to protect the heading, but they are worked off when the last stall of the heading is finished. The chief disadvantage to these methods of working is the formation of detached pieces of goaf, from which gas may issue on to the main roads. There is also much liability to heaving, and a frequent loss of small pillars of coal.

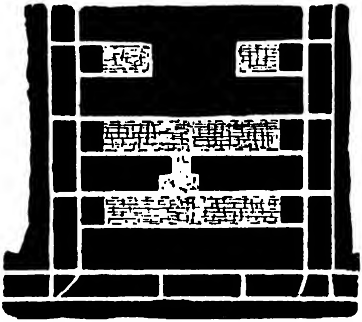


Fig. 118.—Double-stall Method of Working Coal

Wicket Working.—This is a system of working peculiar to North Wales. It is carried on in a manner very similar to the double-stall method. It is applied to seams from 6 to 10 feet thick having little inclination. It is usually attended with much waste of coal.

The Working of Thick Seams, Thin Seams, and Highly-inclined Seams.

SQUARE WORK.—Very thick seams are difficult and often dangerous to work, and usually much of the coal in them is lost. Special methods of working to suit the

circumstances of each case must therefore be devised. The square work method is adopted for working the Dudley Thick, or Ten-yard Coal, in South Staffordshire, a seam ranging from 25 feet to 35 feet in thickness.

Levels, termed *gateroads*, are driven in the lower part of the seam from the shaft towards the coal to be worked. Out of the gateroads a pair of narrow drifts is started, one in the lower part of the seam to open out the square and act as an intake and haulage road, and the other in the higher part of the seam, near the roof, to serve as a return. The intake drift is termed a *bolt-hole*, and is driven 8 yards from the gateroad, and then stalls are opened out from it right and left to form a square, or "side of work" as it is called. The stalls vary from 5 yards to 10 yards in width, and form a side of work when finished from 50 yards to 80 yards square. From six to eight pillars, ranging from 8 yards to 20 yards square, are left in working to support the roof. As the stalls advance, holings termed bolt-holes are made near the roof into the return to ventilate the square, as shown in fig. 119.

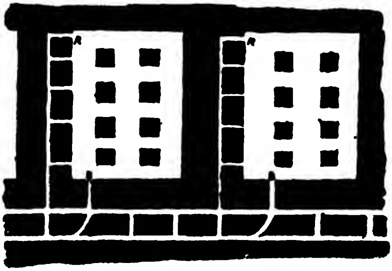


Fig. 119.—Square Work practised in Staffordshire

B B, Bolt-holes. R R, Holings into return.

To work the upper part of the seam, the men stand upon the coals already loosened, or upon scaffolds. When a square of work is finished or abandoned everything is brought out, and the bolt-holes are then hermetically sealed up by a dam built in each place. Fires are very frequent, the seam being very liable to spontaneous combustion. The conditions are very favourable for these fires; much small coal is left in the squares, and it be-

comes mixed with iron pyrites, &c., occasioning frequent spontaneous ignition.

The squares are kept isolated from each other by coal barriers, termed *fireribs*. When the roof will allow it, the pillars of coal standing in the squares are thinned before the side of work is abandoned, but still a large amount of coal is entirely lost. It is said that, owing to the small coal thrown back and the pillars and barriers left standing, not more than one-half, and in some instances, only about one-third of the available coal has been got.

Thick seams in Poland and Bohemia are worked by a modified bord and pillar, and some others by longwall. In Bohemia a seam of coal about 12 yards thick is worked. Large pillars of coal are formed by driving places, 6 feet high, in the lower section of the seam. In removing the pillars the coal is first extracted in this lower portion, so as to undermine the upper. A safety rib of coal, 2 or 3 yards in width, is left during working, and timber is also put in. This rib is taken out, if possible, and the timber drawn, when it is desired to bring down the upper portion. It is often necessary to fire heavy shots by electricity to bring down the huge mass of coal.

THIN SEAMS.—Thin seams of coal are generally more advantageously worked by the longwall or the double-stall methods. Seams down to 15 inches in thickness are at present being worked by the longwall method.

HIGHLY-INCLINED SEAMS.—These are usually difficult to work. The method generally adopted is longwall, with the gateways advancing in steps of 10 yards to 20 yards in width in the line of the strike. The line of the face is, therefore, in the direction of the dip, and the coals loosened at the face slide down to the gateway, where they are filled into the tubs. When the seam approaches the vertical, the coal may be worked in a series of steps one above another, the lowest step in advance of the others. A level is driven in the coal, with the roof and

floor of the seam forming the sides, and the coal forming the roof and floor. Small vertical roads or staples are kept open through the goaf, down which the coals are lowered from the upper steps. It is similar to the method of overhand stoping, with the levels and winzes, of metal mines. The whole method is attended with difficulty, and the cost of getting is very high.

Gob-fires.—Fires arising spontaneously in gobs occur frequently in some mines. Some coal-seams appear to be particularly liable to such gob-fires, and others not at all. The cause, in some cases, may be due to iron pyrites, which, when mixed with small coal, especially if moisture is present, in decomposing generates heat, and may cause ignition of the small coal and other material in the gob. The presence of iron pyrites, although very favourable to spontaneous combustion, is not altogether necessary. It is now thought very probable that many fires arise owing to the oxidation of the coal itself. Heaps of coal on the surface, containing a very small percentage of iron pyrites, have been known to take fire spontaneously; the cause, no doubt, being due more to the oxidation of the coal by absorption of oxygen from the atmosphere than to the presence of iron pyrites. Heat may be generated in gobs also by pressure due to movement of the roof.

The first indication of the presence of spontaneous combustion is given by a peculiar smell termed " gob or fire stink ". An increase of the mine temperature takes place, and vapour and smoke may follow, and afterwards red heat and flame. Such fires may be prevented to some extent by not allowing any iron pyrites, small coal, or any refuse likely to generate heat to be thrown into or left in the gobs. Where a fire is likely to occur as indicated by the thermometer, a large volume of fresh air should be kept circulating along the gateways and gob edge to keep them cool, and to carry off any excess of heat generated before it can rise to such a temperature as to result in fire. When a fire has actually commenced

it is sometimes immediately attacked with water, or an attempt is made to dig it out, and the volume of air passing is reduced; or the district is dammed off, and all access of air closed. A fire is a dangerous occurrence in any mine, but more particularly so where inflammable gas is generated.

CHAPTER XV

COAL-CUTTING; UNDERGROUND CONVEYORS

Under-cutting in Coal-seams. Types of Coal-cutters for Main Roads, Bord and Pillar, and Longwall. Blackett's Conveying System for Longwall.

Under-cutting in Coal-seams.—The most difficult and arduous work of the coal-miner is to undermine the seam in his working place by cutting a groove or channel near the bottom to make a clearance, so that the coal may be broken down by wedges or explosives. This is known as holing, under-cutting, or kirving. The principal object is to obtain as much of the coal as possible in large pieces, as the small coal is of less value. The miner is usually required to hole or kirve along the floor; sometimes, however, the under-clay is soft enough to admit of the holing being done in it; and sometimes a "band" or layer of stone in the seam may be holed. This secures a larger percentage of round coal.

A good workman can hole to a depth of 3 feet with a cut of 16 inches at the front, tapering to an inch at the back of the holing, and for a distance of about 6 yards along the face depending upon the hardness or otherwise of the coal (see fig. 112).

The holing in the case of bord and pillar workings is done along the face from side to side of the working, and along the face in the case of longwall.

As the work of holing proceeds it is necessary to put in supports to stay the coal until the miner has completed his work (see fig. 112). Sometimes as soon as the supports are knocked out the coal breaks off at the back of the holing and tumbles over; in very hard coal it is, however, generally necessary to resort to wedging or blasting to force it down.

The amount of coal left after holing to break or blast down naturally depends upon the height of the seams; in thin seams, say under 2 feet 6 inches, the amount is not more than 60 per cent, the other 40 per cent having been made small in the process of holing. The small coal produced is therefore relatively larger as the height of the seam decreases.

In order to produce a larger percentage of round coal, coal-cutting machines have been introduced, which under-cut the coal and replace hand labour. It is many years since coal-cutting machines were first tried, but only in recent years have they been adopted to any great extent. Each year more machines are put into operation, and the output from them is therefore gradually increasing.

The advantages of coal-cutting machines over hand labour may be summarized as follows:

1. The cost of holing or kirving is less, and in many cases the cost of production is also less.

2. There is an increased production of coal per man employed in the mine.

3. Fewer men are required at the coal face owing to the production being greater, and there is a proportionately less liability to accidents.

4. There is an increased percentage of large coal.

5. There is a more rapid and regular advance of the face of coal, which greatly assists the control of the roof, and minimizes the cost of timber,

6. The straight line of longwall necessary for machines admits of a more regular subsidence and settlement of the overlying strata,

7. Reduced cost of blasting down coal where such is necessary.

8. Smaller cost for timber due to rapid advance of coal face.

9. Reduced cost of keeping up roads as output is obtained from a smaller area.

10. More rapid development of the mine and hence an earlier return on capital outlay.

Coal-cutting machines are seldom advantageously applied under the following conditions:

1. In coal-seams which are soft and so easily wrought by hand labour that holing is unnecessary.

2. In coal-seams which are intersected by many faults at comparatively short distances apart.

3. In coal-seams of exceptional thickness, or where the roof stone is very bad.

Types of Coal-cutters.—The various kinds of coal-cutters may be classified as follows:

For opening out Main Roads ..	Heading Machines.
For holing in Bord and Pillar or Stall Workings	Pick or Percussive Machines. Chain Machines.
For holing in Longwall	Disc Machines. Bar Machines. Chain Machines.

For opening out Main Roads.—Where it is desired to open out main roads quickly in a thick seam of hard coal, machines which cut a circular groove in the face of the excavation are sometimes employed. These machines are termed heading machines. The best-known type, which has been successfully used at Nuneaton Colliery, in Warwickshire, is *Stanley's Heading Machine*. It makes a circular groove in the coal, and when the circular block thus formed is removed, forms a tunnel or heading of from 5 feet to 7 feet in diameter. The machine has a revolving head fitted with projecting arms, upwards of 2 feet in length, supplied with cutters which cut a groove about 3 inches wide (see figs. 120

and 121). The boring head is automatically advanced as the cutters work. As soon as the arms have advanced to the limit of their length, the machine is removed to allow the centre piece of coal to be broken down and filled into tubs. Then the machine is again brought forward and the process resumed. The machine is driven by compressed air, the exhaust of which helps to ventilate the heading.

For holing in Bord and Pillar or Stall Workings.—Very few machines are used in the working places

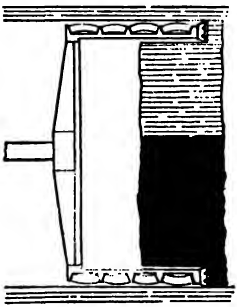


Fig. 120.—Stanley's Coal Heading Machine, showing Revolving Boring Head, with Cutters

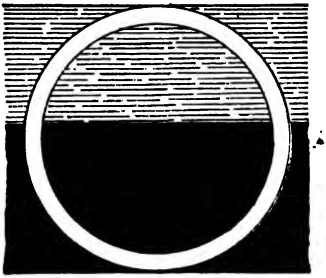


Fig. 121.—Circular Groove cut by Stanley's Machine. The centre is blasted out, and the sides squared every two feet of advance

of the bord and pillar or stall methods in this country, but the number is increasing; they are, however, extensively adopted in the United States.

The pick or percussive machine on wheels was first introduced in 1887. The wheels are usually about 16 to 20 inches diameter, and the machine weighs from 500 to 750 pounds. It is easily moved from one working place to another. In operation, the machine is placed on a wooden platform, 8 feet long by 3 feet wide and 2 inches thick, which is inclined towards the face of coal. The operator sits behind the machine, which he holds by handles, and directs the blow of the

pick or drill against the coal. The pick is reciprocated backwards and forwards, and gives 150 to 250 blows per minute, by means of compressed air at a pressure of 40 pounds per square inch. The cut is made close to the floor of the seam, the depth at the front being 12 to 14 inches, and tapering for 4 or 5 feet to 3 inches at the back. It is considered good work for one machine to cut or hole a length of 20 feet and 5 feet under in an ordinary shift.

The machines of this type best known and most largely used are the Sullivan, Ingersoll-Sergeant (see fig. 122), and the Harrison.

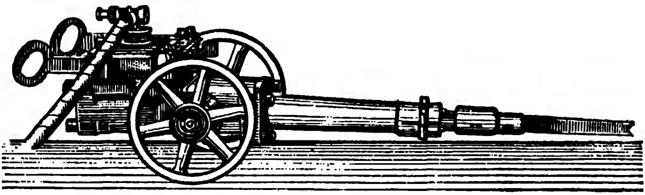


Fig. 122.—The Ingersoll-Sergeant Coal-cutting Machine

The chain machines, when used in bords and narrow headings, are sometimes called breast machines. They are mounted on frames, and in consequence are heavy, and this renders their transport from one working place to another a difficult matter. The chain is endless, and is fitted with cutters which cut a horizontal groove at the bottom of the beam immediately above the thill. Sometimes they are air driven, but very often they are electrically driven.

For holing in Longwall. Disc Machines.—In this country the majority of coal-cutters are employed on longwall faces, and are of the disc type. In this machine the motor, gear box, and haulage gear are made in separate housings which are afterwards bolted together. These parts are so constructed that the bottom forms a flat plate on which the machine slides or skids when

propelled forward. The cutting tools are carried on the periphery of a wheel or disc of from 3 feet 6 inches to 6 feet diameter. This cutting wheel, as it is called, is carried by a suitable bracket or jib plate which projects from the side of the gear box. The wheel is furnished with toothed pockets into which a spur wheel is geared, the whole being driven by means of an electric motor or compressed-air engine through a train of toothed gear to obtain the necessary speed reduction. In practically all modern coal-cutters this train of gear is totally enclosed and runs in an oil bath. To prevent the oil from passing into the motor when the machine is cutting on a steep inclination, some form of oil thrower or oil-retaining device has to be placed between the motor and the gear case. The feed is obtained by means of a small drum driven by a ratchet gear, and carries a length of wire rope which passes to a prop fixed firmly between roof and floor. The rope is passed round a wheel, which is attached to this prop, and then brought back and fixed to the machine. When the motor is switched on, the cutting wheel, which carries the necessary picks or cutters in pockets on its periphery, revolves. The machine is at the same time dragged forward by the haulage rope, and cuts a narrow channel under the coal. Cutting may be done in the coal itself, or it may be done in the underlying material forming the floor. Where the coal is firm and stands up without support until the machine is clear, this type of machine gives excellent results. Should the coal be soft and friable and liable to fall readily, the bar or chain type is better suited for dealing with it. No coal-cutter of any type can operate successfully unless the line of face is straight, and the very first thing that has to be done when a coal-cutter is put in is to prepare a straight line of face for it to work on.

In setting out the line of face it is necessary to consider whether the coal should be worked " end ", " half-end ", or " face ", that is, whether the machine should

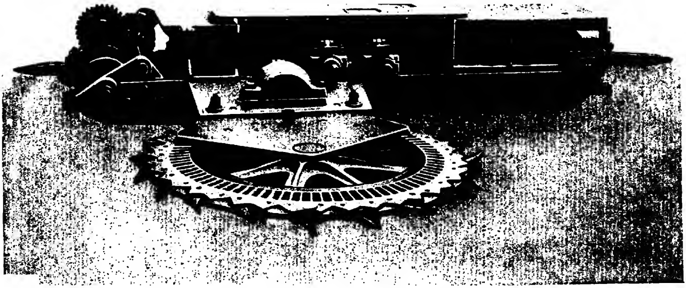


Fig. 123.—Anderson Boyes Disc Machine

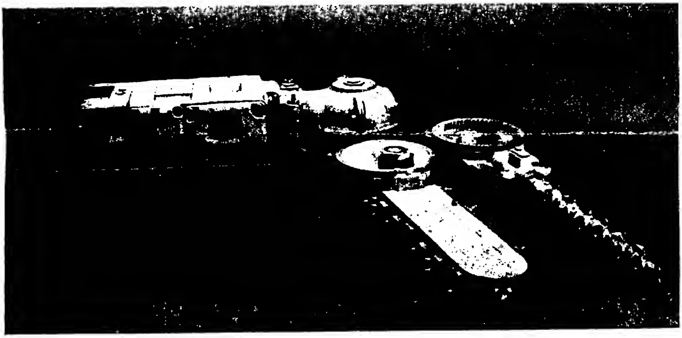


Fig. 124.—Mavor & Coulson's Bar Machine, Universal Type

Fig. 125.—Anderson Boyes Chain Machine

cut across the main lines of cleat or cleavage, or along the line of cleat. The nature of the coal usually determines this. In a strong coal it will probably not matter which way it is worked, but in a softer coal, or one which is liable to break off in slabs, it is better to work it on " end ", otherwise the coal is apt to come over upon the machine and men.

The Anderson Boyes Machine.—This machine is largely adopted. It is of heavy construction, and is specially arranged for making deep cuts, either in hard coal, or in the stone immediately underlying. The principle of the construction is that already described.

The machine which is shown in fig. 123 is of the totally enclosed type, all gears being run in oil, which greatly reduces the noise while the machine is operating. The motor and all electrical parts are in flame-proof cases, so that in event of gas being ignited inside, the flame of the explosion will not pass out and ignite any external accumulation that may be present at the moment of ignition. The design shown is 16½ inches high, and can be fitted with either an A.C. or D.C. motor. The motors are of compact design and robust construction, so as to develop the maximum power possible for the space available. The starter is of the drum type with cast-iron grid resistances. All insulation is of mica. The gear case is of cast steel with all bearing brackets formed on the inside; the gears are designed to occupy the smallest possible space and are run in oil. Provision is made for the continuous lubrication of the cutting wheel while in the cut, a feature which saves much valuable time and prevents undue wear of this important but not always readily accessible part. This is the favourite type of disc machine in the Scottish coalfields, where coal-cutting is more highly developed than in any other part of the United Kingdom.

BAR MACHINES

In machines of this type the cutting tool consists of a round bar at right angles to the frame, slightly tapered towards the outer end, and fitted with cutters. The bar is made to revolve rapidly, and in this way cuts a groove along the face to a depth of from 3 feet to 6 feet according to requirements.

The advantages claimed for the bar machines are:

1. The machines do not occupy so much space as the disc machines, and therefore timbers can be set nearer to support the roof.

2. The tapered form of the bar makes a cut tapered from the front, and this favours the fall of the coal and allows of it being more readily broken up to the required size for filling up.

3. The bar is more adaptable, as it can be raised, lowered, or inclined, and can be swung round in or out of the cut, and it can cut its way into the coal at starting.

The disadvantages of the bar machines are:

1. They will not make a cut on the floor level, but leave a few inches of the seam on the floor which has to be taken up by hand afterwards.

2. They do not clear out the cuttings so readily as the disc machines.

3. In a stratified material they cut across the beds vertically instead of horizontally.

An example of a bar machine is:

The Pickquick Bar Coal-Cutter.—This machine is usually electrically driven. On the frame is carried a motor, connected by spur-gearing to the cutter bar (see fig. 124), which not only is made to revolve, but has a reciprocating motion of about 2 inches within the cut. The advantage of this combined rotary and reciprocating motion is to give a shearing and chipping action, and enable the bar to better clear the cut of the debris. The bar runs at about 420 revolutions per minute, the

ratio of gearing with the motor being 2 to 1, and for a 5-feet cut the bar is made 7 feet 8 inches long. The bar can be swung round horizontally through 180 degrees, so that it can cut its way into the coal or other material, and it can be tilted upwards or downwards. In recent designs the bar and the lower part of the gearhead can be removed, and a corresponding part carrying a jib and chain substituted, thus converting the bar machine into a chain machine. The machine is made in three sizes, namely:

For cuts up to $3\frac{1}{2}$ feet;	weight, 20 cwt.;	motor, 12 horse-power.
" " $4\frac{1}{2}$ "	" 30 "	" 18 "
" " 6 "	" 45 "	" 26 "

Chain Machines.—During recent years, machines have been introduced fitted with a cutting arm or jib around which a chain carrying cutters is made to revolve and cut into the coal in a similar manner to the disc.

The chief advantage appears to be that the chain brings out the cuttings, and as it works at the end of the frame, the cuttings are left behind, and can be readily dealt with. The chain makes its cut along the floor level, and has the same horizontal movements as the bar.

An example of a chain machine is:

The Anderson Boyes (fig. 125, p. 168).—This coal-cutting machine is electrically driven. The cutting arm or jib is pivoted at the end of the machine, and the chain carrying the cutters is driven by a sprocket-wheel gearing with the armature shaft. The jib can be slewed round through more than 180 degrees, and will cut on either side of the machine, which is reversible, and can cut its way into the coal.

The chief feature about this machine is the chain and the disengaging gear. The chain is run in guides, which ensures that the links carrying the cutting tools cannot crank up under heavy load. This is a decided

advantage, as it keeps the cutting tool always at the right cutting angle irrespective of the load. The driving gear can be disconnected from the chain and the motor run with the chain remaining stationary. The jib is swung into or out of the cut by power, a powerful jib head being provided to stand the stress of this operation. Only single picks are used, and are fitted into pockets in the chain at the proper cutting angle. The machine can be fitted for either alternating or direct current, and every precaution to render the various parts flame-proof is taken. For working under a bad roof the chain machine has no rival. In the case of the machine described, its entire length is 7 feet 8 inches and its breadth 2 feet 0¾ inch. This feature allows the roof to be secured close up to the face of the coal, while still leaving room for the machine to operate. Like practically all of the modern types of coal-cutters the machine cuts at floor level, and skids along the floor. A chain and sprocket wheels are used for propelling the machine forward, in place of the more usual rope.

Conveyors.—These appliances are used for conveying the coal from the working face to the roadway, where it is loaded into tubs. The method is only suitable for longwall working, and then only in certain suitable seams, but where conditions are favourable it saves the miner much arduous work and prevents breakage of coal. Conveyors may be worked in conjunction with coal-cutters, but in many instances they give the greatest satisfaction in thin seams which are easily worked by hand. They are usually worked on a face of 100 to 150 yards in length, with a main road from which the coal is drawn in the centre of the face. The ventilation is carried in by a roadway at one end, and, after passing along the face, out by another road at the opposite end. If the formation of these roadways does not supply a sufficient amount of packing material, it must be drawn from the waste, as nothing is more detrimental to the success of the system than having the roof

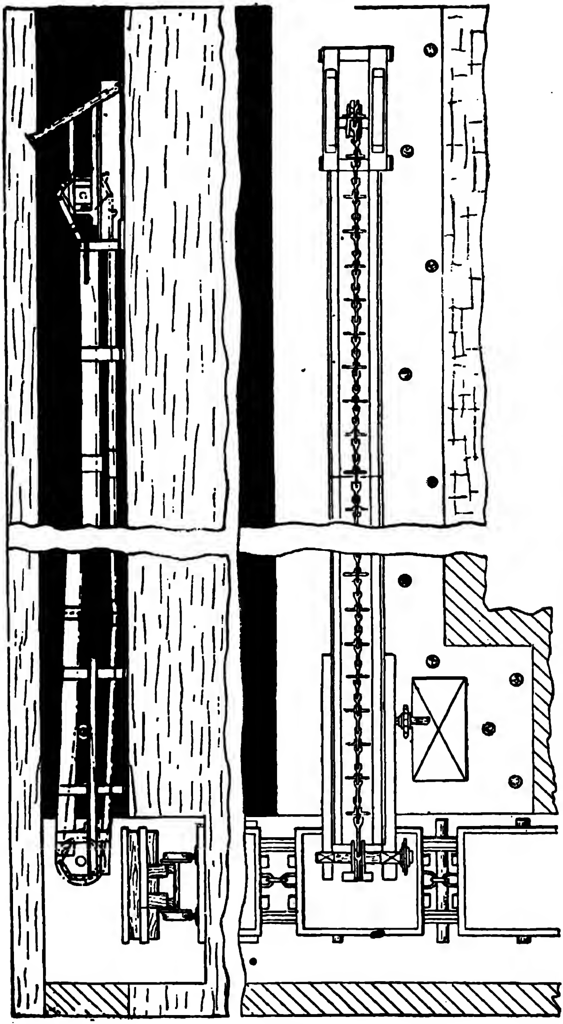


Fig. 126.—Blackett's Patent Underground Conveyor System

collapse along the working face, as it will ultimately do unless it is properly packed. Conveyors may be divided into three classes:

1. Belt conveyors.
2. Sliding carriage conveyors.
3. Shaking conveyors.

An example of the belt type is:

Blackett's Conveyor for Longwall Workings.—This system is of comparatively recent invention, and has successfully passed its experimental stage, as its practical utility has been proved by its use over a lengthened period.

Its object is to admit of gateways in longwall workings being placed much farther apart than is the case when coal is loaded into tubs in the ordinary way, and so reduce the cost of production.

The conveyor, which consists of an endless chain belt running in a flexible metallic troughing, is actuated by a motor, specially designed, of 8 to 10 horse-power, and placed parallel to the face of the coal (see fig. 126).

The coal having been under-cut by a coal-cutter, and broken down, is filled by hand labour on to the conveyor, which carries it along to the gateway, where it is automatically tipped into tubs.

The conveyors have been made up to 100 yards in length, which means that the gateways, instead of being 10 or 15 yards apart as in the ordinary course of things, may be 100 yards apart. The cost of making and maintaining gateways, except at 100 yards apart, or whatever distance less than this is decided upon, is therefore dispensed with, and thin seams of coal, which might otherwise be worked at a loss, may possibly be profitably worked by this arrangement.

Among the advantages claimed for this system are:

(*a*) A reduction in the number of gateways in longwall.

Fig. 127.—Thomson's Conveyor

To face page 174]

(*b*) Greater facility in filling coal which has been under-cut.

(*c*) Rapid and regular advance of the coal face.

(*d*) The "putting" or "tramming" distances are reduced.

Thompson's Conveyor.—This conveyor is formed from a number of sledges or pans about 18 inches wide, 6 feet long, and fitted with a plate 9 inches high on one side, the other being left open. These pans are fastened together so as to form a train, which is drawn from end to end of the face by means of a wire rope driven by a motor. The road is formed by ripping the floor, and a bridge is placed over the gap thus formed, and, as the pans containing the coal pass over, the coal is shed by means of a sloping plate into the tubs which are placed beneath. The whole appliance is easily moved forward as the face advances, and gives, under suitable conditions, very good results. The method of loading is shown in fig. 127.

Summerlee Conveyor.—This conveyor consists of a series of light iron troughs or trays 20 inches wide, 3 inches deep, and 6 feet long, joined together by a suitable locking device. The whole is hung by chains from cross-bars set up to the roof, and given a quick forward motion with slow return by means of elliptical gear driven by a motor. The coal is shovelled into the trough and passes along it because of the reciprocating movement, being discharged over the end into the tub. It is best suited for working in seams of average thickness, and can, by reversing the direction of the movement, be used for carrying rubbish from the road along the face for packing purposes.

CHAPTER XVI

EXPLORING DRIFTS; DAMS

Driving and Boring Exploring Drifts in Coal-seams towards Old Workings, likely to contain a Dangerous Accumulation of Water or Gas. Burnside's Safety Boring Apparatus. Mine Dams.

Precautions Required.—The Coal Mines Act, 1911, enacts that " Where any working has approached

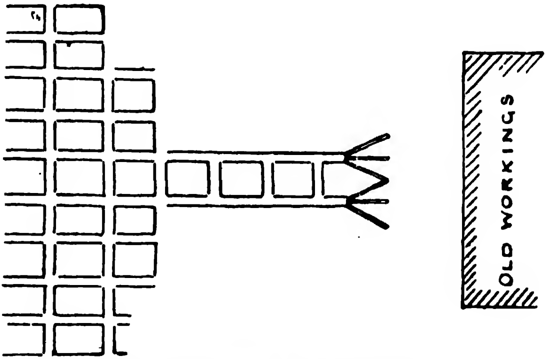


Fig. 128.—Exploring Drifts advancing towards Old Working

within forty yards of a place containing, or likely to contain, an accumulation of water, or of disused workings, the working shall not exceed eight feet in width, and there shall be constantly kept extending to a sufficient distance, not less than five yards in advance, at least one bore-hole near the centre, and sufficient flank bore-holes on each side at intervals of not more than five yards ".

In the plan shown in fig. 128 the working places are advancing towards old workings which contain water or gas in sufficient quantity to prove dangerous if holed into unexpectedly, consequently it is considered a necessary

precaution to have a pair of leading-places or exploring drifts carefully bored going direct to the old workings, considerably in advance of the regular working places.

The clause of the Act already quoted stipulates that the exploring drifts shall commence at least 40 yards from the place containing the dangerous accumulation. It is, however, not considered prudent to defer boring until the surveys show a thickness of only 40 yards of coal to the old workings, unless there is absolute certainty of the exact position of the faces of the two sets of workings.

The plans of ancient workings are so frequently incorrect and incomplete that several disastrous inundations have occurred through unexpected holings into old workings, owing to the plans indicating a more remote position of the latter than is the fact. A case of this kind occurred in the Maudline Seam at Newbottle Colliery, in the county of Durham, on the 24th June, 1885, whereby thirteen men met their death by the noxious gas which arose from an unexpected holing into some old workings, causing an outburst of gas and water. No exploring or boring-place had been commenced because the plans showed the face of the nearest workings to be 376 yards distant, but the unfortunate holing proved the old plans to be inaccurate to that extent.

Arrangement of Boring-places.—When boring is commenced at a considerable distance from the supposed position of the old workings, it is necessary to have a pair of places going together in order to get the face ventilated. An example is given in fig. 129. This sketch shows a pair of exploring drifts and the position of the bore-holes, which are bored in the face and always kept in advance. This method of driving and boring is not always a safe one; in deep pits, and especially in tender coal, it is essential to leave a considerable width of pillar between the drifts, and this gives rise to the possibility of the flank or slope holes being so far apart

between the drifts as to allow one of the old workings to get between the holes undetected. In fig. 129 the flank bore-holes between the drifts are shown to meet each other, and so long as they do this the drifts are safe, but when the bore-holes are short and leave a rib of unbored coal, then the system is unsafe.

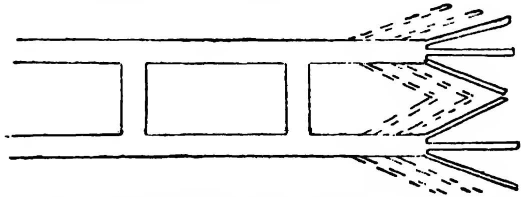


Fig. 129.—Method of Boring both Exploring Drifts

A method which is more frequently adopted, and is perhaps safer than the foregoing, is illustrated in fig. 130. One of the pair of drifts is bored with three holes, and kept a short distance in advance of the other one. Each stenton is bored with three holes and driven from the

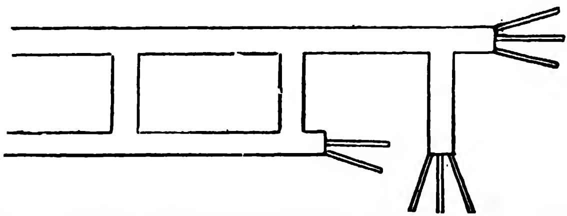


Fig. 130.—Another method of arranging Bore-holes in the Exploring Drifts

leading drift; when it is a sufficient distance, the other or back drift, which has been standing at the previous stenton, is commenced, and is bored with one front and one flank hole, the latter on the side of the solid coal. In this way it is driven up until it holes into the stenton, and so completes another winning.

When the plans and other circumstances indicate that the old workings are very near, and especially if a heavy pressure of water or gas is anticipated, it is usual to drive the fore or advance drift forward about 8 feet in width, and ventilate it with a wood or brick brattice. This method is illustrated in fig. 131. Two front or straight-on bore-holes, and the usual flank holes, are kept in advance, and they are all, in most cases, increased in length as an additional precaution. Sometimes holes are bored at right angles to the drift to ensure a sufficient thickness of side coal, for it might happen that an old

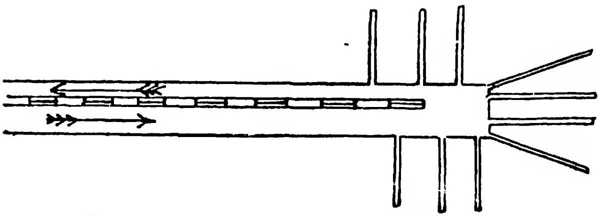


Fig. 131.—Arrangement of Bore-holes in a Single Drift

working place was running parallel to the exploring drift, but not near enough to be holed into by the flank holes.

Boring Appliances.—The boring of the holes is sometimes done by means of light iron bore-rods, similar in construction to those used for vertical bore-holes; they are 6 feet long, with a chisel to bore a $1\frac{1}{2}$ or 2 inches hole. A bracehead is screwed on to the end, and two men work them backward and forward in the hole to give the stroke, and at the same time they give them a partial turn so that the chisel will bore a circular hole.

The work, however, is more often done by boring-machines, of which there are many kinds. These accomplish the boring much more easily and in less time than do hand-borers. The price paid for boring is usually about sixpence per yard.

Position of Bore-holes.—The bore-hole at the centre of the drift is called the front or straight-on hole, and is bored in the same direction and at the same inclination as the drift is being driven. The length of the hole must not be less than 5 yards in advance, according to the terms of the Act, but a greater length than this is often necessary owing to the depth from the surface, the pressure anticipated in the old workings, and the nature of the coal.

The bore-holes near the sides or nooks of the drift are termed the *flank* and sometimes the *slope holes*, and are bored at an angle of about 30° from the line of the drift. They are bored up the same distance as the front hole, and fresh ones are put in with every advance of 2 yards of the drift. The drift must not be more than 8 feet wide in any part of it; a more convenient and safer width, however, is 6 feet, with the front hole in the centre, and the flank holes each 2 feet 6 inches from it. A template is usually provided in order to keep the flank holes at the required angle.

Precautions when Boring.—When boring towards old workings the following precautions should be taken.

Three or four fir-wood plugs ought to be kept dry and in readiness at the face of each drift. They are usually made from 4 to 6 feet in length, and tapered at one end almost to a point for easy entrance into the bore-hole when there is an outflow of water, the other end being made larger than the diameter of the bore-hole, and hooped with iron to prevent the wood splitting when it is being driven into the hole. The driving of a wood plug into a bore-hole against a heavy pressure of water is sometimes both difficult and dangerous. A cross piece is usually put through the plug near the hoop to serve as a handle; and the plug may also be lapped with flannel and tar.

A set of *trying* or measuring-rods should always be kept at the face, so that the length of the bore-holes can at any time be ascertained. This is necessary in order to

avoid anything less than the minimum or statute limit of 5 yards of bore-hole being kept in advance of the drift.

Safety lamps should always be used while boring, and a spare safety lamp kept burning a short distance back from the face, so that if the borers' lamps become extinguished by a sudden outburst of water or gas on holing into old workings, there will be another at hand.

Sometimes when heavy pressures of water and gas are anticipated, boring is accomplished through a fixed pipe and valve at the mouth of the hole, and when a holing is effected and the rods withdrawn, all that there is to do to prevent the outflow of water or gas is to close the valve. Also a gauge may be fixed on the pipe to ascertain the pressure of the water in the old workings.

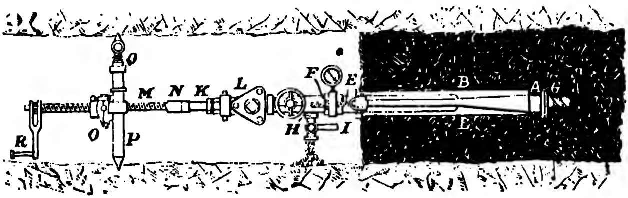


Fig. 132.—Burnside's Boring Apparatus

Burnside's Safety Boring Apparatus.—This is designed to effect the safety of boring long-holes against old workings, in which water and gases may have accumulated under great pressure. Before commencing to bore, the machine is tested to a water pressure of 600 lb. per square inch. An important feature of the apparatus is the provision made for shutting off the water and gas as soon as a holing is effected, and in having the latter under control to liberate as required. The first operation is to bore an ordinary hole about 2 inches diameter to a distance of 4 feet into the coal. This is then enlarged to 4 inches diameter to a distance of 3½ feet. A tube A (fig. 132), and two iron wedges B, B, and two supporting plates are inserted into the drill-hole, the

wedges and plates being made to fit round the tube. The outer ends of the wedges are attached to the cross-head flange by two bolts E, E. When the nut F is screwed up, the two wedges B, B are drawn outwards, and as a result the supporting plates are locked, and the india-rubber washer G at in-end of tube becomes tight, and renders the tube gas- and water-tight. The pipe marked H, which is closed with a gun-metal tap I, is to allow the debris from the boring-tool to fall on to the floor of the mine. The boring-rods are inserted inside the tube to reach the inner end of the piston-rod K, which works through the stuffing-box L, and is screwed on to the outer end of the tube. The feed-screw M is attached to the outer end of the piston-rod K by means of the coupling N. The feed-screw M is worked through the feed-nut O; this is secured to the upright stand P, which in its turn is attached to the roof and floor of the seam by means of the tightening screw Q. The handle R is operated upon by manual power, and the boring-tool, the boring-rods, and the piston K are propelled by the feed-screw M, and so the hole is bored. The boring-rods are twisted in the form of a spiral to bring the debris out of the hole to H, where it falls through the tap I and discharges itself.

DAMS

These are frequently required to be built in mines for one of the following purposes:

1. To divert the direction of feeders or streams of water.

2. To shut off water entering abandoned workings in order to avoid the expense of pumping.

3. To exclude air and smother fire in places where fires arise underground.

Construction of Dams.—To divert the direction of feeders of water, straight dams built of wood and clay, or of brick, are generally adopted, this being

sufficient to resist the small pressure of the water. A wood and clay dam is constructed of two parallel rows of battens or balks, placed edge upon edge from 1 to 2 feet apart, and fitted into grooves cut on both sides of the drift. Between the two rows of timber, clay is beaten very tightly in, so as to make the dam water-tight (see fig. 133). For the purpose of shutting off water entering abandoned workings, the structure must be sufficiently strong to resist the pressure of water due to the depth of the workings, and must be completely water-tight. Straight brick dams are not recommended for resisting great pressures; wedge-shaped dams of wood, or cylindrical dams of bricks and cement, are stronger and more often adopted, the former being preferred on the Continent and the latter in this country.

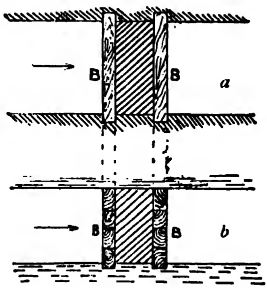


Fig. 133.—*a* is a plan of straight wood Dam. *b* is an end view. B B, Battens or balk with clay between. The arrow indicates direction of water pressure.

In the wedge-shaped dam the broad end is always put against the pressure. The timber used is cut to the proper size and shape and dressed on the surface, each piece being numbered before being sent into the mine, so that they may be put together accurately in the site prepared for the dam. Much leakage along the joints of the pieces of timber may take place for some time after the water comes against the dam, but if the wood used be perfectly dry when put in, it swells with the moisture, and afterwards becomes completely water-tight. One advantage claimed for the wooden, or *frame-dam* (see fig. 134), as it is sometimes termed, is that it is not liable to crack with any movement of the surrounding strata, which is the case with brick dams. Fig. 135 shows the plan of the cylindrical-shaped dam. Dams of this form

have been built to withstand great pressures of water. The bricks should be carefully selected, the most suitable being hard burnt, which will not absorb much moisture. Portland cement is generally used to lay the bricks. This should be laid on evenly but not too thickly, and should be quick-setting.

Position.—The site of dams should be most carefully chosen. The vicinity should be free from faults, fissures, or other irregularities, and the beds impervious. No

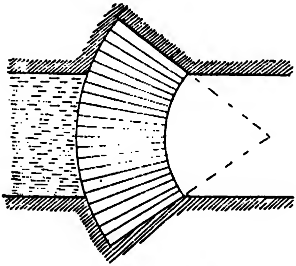


Fig. 134.—Frame on Wood Dam

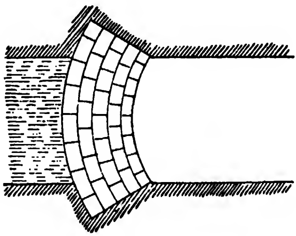


Fig. 135.—Brick and Cement Dam

shots should be fired in preparing sides, roof, or thill to the required shape; they should be prepared with picks and wedges. Sometimes these surfaces are so rough that a coating of cement is laid upon them, upon which the bricks are laid.

During the building of a dam provision must be made for the escape of the water. This is generally done by building into the dam, at the bottom, a pipe or pipes of sufficient diameter to allow of the passage of the feeder of water. When the dam is finished, a wood plug, placed on the inside of the dam at the commencement, is drawn into the pipes by means of a rod passing through the pipes to the outside. Sometimes these pipes are afterwards filled up with cement. Another plan is to fit a valve on to the pipe end, which can be closed when the dam is completed. A second pipe of small diameter, fitted with

a valve at the out-end, is usually built into the dam, close to the roof, to allow of the escape of all air and gas as the water accumulates behind the dam. A pressure-gauge is sometimes screwed on to this pipe to denote the pressure of the water.

To Extinguish Fires.—This operation is attended with extreme danger to those engaged in it. Sometimes the fires are flooded, and temporary stoppings are erected to exclude the air and smother the fire. If the enclosure be not complete, the air passing through will allow the fire to spread; if the enclosure be tight, the gases may be forced through the crevices and endanger the workmen employed. If fresh air be allowed to pass to carry off the gas, it may give rise to an explosion by mixing the firedamp (CH_4) with the proper percentage of air.

The chief object is to exclude the air from the fire, and in building the dams it is necessary they should be made air-tight. In Staffordshire and Warwickshire, where gob-fires are frequent, brick-and-mortar dams are frequently required to isolate a district where a fire has commenced, from the rest of the mine, so that the fire may die out.

In some mines continuous walls of clay or sand, called *wax walls*, are built by the sides of the main roads to prevent air from getting into the goaf, and so prevent combustion taking place.

CHAPTER XVII

COMBUSTION

What is Combustion?—Combustion or burning may be defined to be chemical action taking place in a substance by which light and heat are evolved. This action consists in the chemical combination of elements,

chiefly carbon and hydrogen with oxygen, the latter being generally derived from the atmosphere. The light produced by the process of combustion may at first be too feeble to be seen by the naked eye, but as the heat or temperature due to the chemical action rises, the light may appear in the form of a ruddy glow, which may continue to increase until white heat or incandescence is reached.

Composition of Combustible Substances.—The combustibility or property of undergoing chemical change by the process of burning belongs to various bodies in very different degrees, depending upon their chemical composition. Those which are composed principally of carbon and hydrogen contain the elements essential for ready and complete combustion. When, however, combustible substances contain proportions of other elements than carbon and hydrogen, the process of combustion may be to some extent interfered with. The substances which are in common use for producing illumination and heat are termed combustibles, and are compounds of the above-named elements. Coal is the most abundant and most largely used combustible for producing light and heat for the various purposes required by man. The average chemical composition of coal is about 80 per cent carbon, 5 per cent hydrogen, 8 per cent oxygen, 3 per cent nitrogen and sulphur, and 4 per cent ashes. Other combustibles are coke, wood, peat, paraffin-oil and other oils, tallow, and other fatty substances. A substance consisting largely or entirely of carbon and hydrogen in combination is known as a hydrocarbon.

Air required for Combustion.—Before combustion can take place in any substance, it must be converted from the liquid or solid state in which it may be to the gaseous. Sufficient heat must be applied to some portion of it to convert it into the form of gas. The carbon and hydrogen are in a state of chemical combination in a combustible substance, and they must be broken up and chemically dissociated from one another. When heat

is applied sufficiently to the substance the carbon and hydrogen become separated, and each unites with oxygen in the process of combustion, forming two new compounds, namely, carbonic acid gas and water. A supply of oxygen is therefore necessary to produce combustion. Air contains 23 parts by weight of oxygen and 77 parts of nitrogen, and because of this proportion of oxygen it is a supporter of combustion, this being the element required for the process.

It is sometimes necessary, in connection with steam boilers and underground furnaces, to calculate the quantities of air required in order to supply sufficient oxygen to support combustion. When the composition of the fuel used, or of any other combustible substance, is known, the exact quantity of air required for its combustion may be calculated.

Combustion does not take place nearly so rapidly in air as it does in pure oxygen, because air contains such a large proportion of nitrogen, which acts as a diluent. In pure, undiluted oxygen, combustion takes place with great rapidity and brilliancy. A fire burns more fiercely when air is forced upon and through it because of the increased supply of oxygen. Advantage is taken of this fact where hot fires are required, as, for instance, in the case of steam boilers, and currents of air are forced through the fires to increase the rate of combustion and the amount of heat.

Products of Combustion.—The products or results of the complete combustion of any hydrocarbon are carbonic acid gas and water. When the carbon and hydrogen are dissociated, part of the oxygen unites with the carbon and forms carbonic acid gas, and the other part of oxygen unites with hydrogen and forms water. When coals are burnt the carbon and hydrogen of the coals unite with the oxygen of the air; light and heat are produced by this chemical process, and the resultant gases pass up the chimney together with some finely divided but uncombined particles of carbon in the form

of smoke and soot. These mingle with the atmosphere, the ashes being left behind. The chemical combination of all gases takes place in fixed quantities both by weight and volume. Thus one volume of carbon always combines with two volumes of oxygen, or 12 parts by weight of carbon with 32 of oxygen, to form carbonic acid gas, and two volumes of hydrogen combine with one volume of oxygen to form water.

Dr. Angus Smith has analysed the gases present in a common house-fire, with the following results:

	Carbonic Acid Gas.	Carbonic Oxide.	Oxygen.	Nitrogen.
Gas from the clear fire, below	16·10	—	4·95	78·95
Gas from a heap of glowing coal ..	18·17	2·48	—	79·35
Gas from the upper part of the fire, 1 inch below surface	20·80	0·99	—	79·21

The nitrogen given in the analysis is for the most part derived from the air; the water formed by the combination of hydrogen and oxygen is not stated.

When fresh coal is added to a fire the temperature is lowered and incomplete combustion results. As a consequence dense black smoke is formed and carried off by the draught of air along with the other products of combustion. The black smoke is so much unburnt carbon, and represents wasted coal and heat.

Degrees of Combustibility.—Some combustible substances burn more brilliantly and with greater rapidity than others. The degree of combustibility varies in the inverse ratio to the chemical affinity which the constituent elements of the substance have for each other; therefore the weaker the bond by which they are held together the more rapidly do they combine with the oxygen of the air in the process of combustion.

Coal varies greatly in chemical composition, and also in combustibility. Some varieties are easily ignited, and burn very quickly; other kinds, as anthracite, are difficult to ignite, and burn slowly, but give off intense heat.

Firedamp is a compound the combustion of which is almost instantaneous. It is composed of one volume of carbon to four volumes of hydrogen, and will extinguish flame in the absence of oxygen. When brought into contact with a certain proportion of atmospheric air, or with free oxygen, and ignited, the chemical combination takes place so rapidly and violently that the combustion of firedamp is then termed an *explosion*. When an explosion of firedamp mixed with air takes place, the whole or part of the oxygen of the air is used up and the nitrogen remains free, the resultant gases, which are termed *after damp*, being composed of carbonic acid gas, water, and nitrogen. When the mixture contains one part of firedamp to ten parts of air the combustion is complete, and is represented by the following chemical equation:

Before explosion.				After explosion.				
CH_4	+	$4(N_4 + O)$	=	CO_2	+	$2H_2O$	+	$16N$
Firedamp	+	Air	=	Carbonic acid	+	Steam	+	Nitrogen
2 volumes	+	20 volumes	=	2 volumes	+	4 volumes	+	16 volumes

Explosives are substances the combustion of which is instantaneous. They contain sufficient oxygen in themselves to complete the process of combustion instantaneously when ignited. Ordinary combustibles, as already explained, are dependent upon the atmospheric air for the oxygen required for combustion, but gunpowder and the various explosives named in Chapter X explode because they contain the necessary elements in their composition, and do not need any supply of oxygen from the outside air.

CHAPTER XVIII

HEAT AS A FORM OF ENERGY; MEASUREMENT OF HEAT; THE THERMOMETER

Heat.—We have already stated that in the process of combustion heat is evolved. The heat derived from burning coal is the result of the chemical combination of the oxygen of the air with the carbon and hydrogen contained in the fuel. Heat is generally produced when chemical action is developed in substances. The spontaneous combustion of coal, the slaking of lime, the warmth of rotting vegetation, are examples of the evolution of heat when chemical action is set up.

Heat is a form of energy which may be manifested through the medium of some material agent. It can be employed to set bodies in motion, to raise weights, or do any other mechanical labour; and, on the other hand, work or energy can be converted into heat. Heat is stored up in greater or less quantities in all combustibles, and by the process of combustion it is evolved, and may be converted into work. Thus, coal contains a large quantity of heat, and this heat is converted into energy for the working of steam-engines. The heat of the fire is communicated to the water and steam in the boiler, the steam carries this heat with it to the cylinder of the engine, where it produces motion of the piston, and enables it to do mechanical work.

UNIT OF HEAT.—In order that the quantity of heat yielded by combustibles may be measured, a standard substance, to which all measurements of heat may be referred, and a unit of measurement, must be decided upon. In Britain the substance selected is water, and the unit of heat is called the British Thermal Unit. It may be defined as the amount of heat required to raise the temperature of a pound of water by 1° F., when at its maximum density, i.e. from 39·1° to 40·1° F.

The quantity of heat yielded by a given weight of any combustible substance may be estimated and measured by the unit of heat. It is only necessary to burn the substance in such a manner that the heat given off may be communicated to a known volume of water, and then the increase of temperature measured. The number of units of heat evolved by the combustion of a pound of coal is often termed its calorific power. The calorific power of one pound of hydrogen has been ascertained to be about 60,000, and of one pound of very good coal about 14,000.

SPECIFIC HEAT.—Equal weights of different substances require very different quantities of heat to produce in them the same change of temperature. This is expressed by saying that different bodies have different specific or relative heats. The specific heat of a body is the ratio of the quantity of heat required to raise the temperature of that body one degree, to the quantity required to raise an equal weight of water one degree.

The units of heat required to raise a given substance a given number of degrees may be found by multiplying the weight of the body by the number of degrees and then by the specific heat of the body.

TRANSMISSION OF HEAT.—By the transmission of heat we mean the various ways in which heat is communicated from one body to another. There are three ways in which heat passes, viz.:

1. *Conduction.*—Conduction is the communication of motion by one molecule to another. All substances do not possess the property of conduction in equal degree. Those which allow of the ready passage of heat are termed good conductors; whilst those in which the passage of the heat takes place slowly are termed bad conductors. Metals, as a rule, are good conductors, silver having the highest conductivity, and copper next; whilst such substances as glass, ivory, wood, wool, &c., are bad conductors.

2. *Convection.*—When water is boiled in any vessel, as

in a steam boiler, the layer of water at the bottom becomes heated first, expands, and rises; while the cooler layers at the surface gradually sink. Ascending and descending currents are thus produced, by which the heat is distributed throughout the liquid. This mode of transmitting heat is called *convection*.

3. *Radiation*.—If we stand in front of a fire or any source of heat we at once experience a sensation of warmth. The burning fuel gives off rays of heat in all directions. This mode of transmitting heat through space is called *radiation*, and is the way in which we receive heat from the sun.

SOURCES OF HEAT.—The chief source of heat is the sun. Another source, which has already been mentioned, is that of chemical action. Other sources of heat are friction, compression, percussion, and electrical action, and the earth itself contains a certain amount of heat.

GENERAL EFFECTS OF HEAT.—As a general rule, all bodies expand with an increase of heat and contract when cooled. Gases expand most, liquids next, and solids least when equally heated.

Gases expand alike and equally for equal increments of heat in the ratio of $\frac{1}{459}$ of their volume for every degree Fahrenheit. This being the rate of expansion, it follows that if we take a given quantity of air and raise its temperature one degree, it will increase in the ratio of $\frac{1}{459}$ of its original volume, and so on for every one degree increase of temperature.

Liquids do not expand so much as gases. The amount of expansion of liquids must be estimated by the increase in volume which takes place when heated in a vessel.

Solids expand and contract by comparatively small amounts. The expansion may be estimated by the increase in length and by the increase in volume. Cast iron expands in length 0·00000617 when the temperature is raised 1° F., and wrought iron 0·00000642. In all constructions where iron and other metals are employed,

as, for example, railways and bridges, allowance must be made for expansion and contraction due to variations in the temperature of the atmosphere.

THE THERMOMETER.—The expansion of bodies by heat and the contraction by cold affords an easy method by which the alterations in temperature which they may undergo may be observed and measured. Some substance is used as a medium for the measurement of the alterations. Fluids are best adapted for this purpose, mercury being in common use for ordinary temperatures and alcohol for very low temperatures.

The scale of measurement of the amount of expansion and contraction is purely arbitrary, and varies in different countries.

Graduation of the Thermometer.—In graduating the thermometric scale advantage has been taken of the physical property of water, which freezes and boils at constant temperatures at normal atmospheric pressure. These two points are fixed upon, namely, the freezing point of water, or temperature of melting ice, and the boiling point, or temperature of boiling water.

To obtain the former or freezing point the thermometer, which consists of a glass tube with a narrow bore of uniform diameter throughout its length and a small bulb or reservoir at the bottom containing mercury, is plunged into melting ice; the mercury sinks in the tube and at last becomes stationary; a scratch is made on the glass at this point. The thermometer is next placed in the steam rising from boiling water. The mercury rises to a certain point and there stands. This is also marked off on the tube, and is called the boiling point. The space between these two points now remains to be marked off into equal parts to complete the thermometer.

Thermometric Scales.—There are three different scales applied to thermometers:

1. Fahrenheit's Scale.—The freezing point of this scale is marked 32° (degrees), and the boiling point 212°,

the space between the two points being divided into $212 - 32 = 180$ equal parts or degrees. The scale is continued downwards from 32° to 0° (zero). The Fahrenheit scale is much used in this country, and is adopted at all collieries.

2. Centigrade Scale.—The freezing point of this scale is marked 0°, and the boiling point 100°. The space between the freezing and boiling points is thus divided into 100 equal degrees. This scale is adopted in France.

3. Réaumur Scale.—The freezing point of this scale is marked 0°, and the boiling point 80°. This scale is used in Germany.

The freezing and boiling points of the three scales are therefore as follows:

	Fahrenheit.	Centigrade.	Réaumur.
Freezing Point	32	0	0
Boiling Point	212	100	80

Conversion of the Degrees.—The readings taken in one scale may easily be converted into the corresponding readings of the other two. In comparing the three scales we have only to notice that the space between the freezing and boiling points in Fahrenheit is divided into 180°; in Centigrade, into 100°; and in Réaumur, into 80°. These numbers are in the ratio of 9 : 5 : 4. Thus:

$$180^\circ \text{ F.} = 100^\circ \text{ C.} = 80^\circ \text{ R.};$$
$$\text{that is,}\quad 9^\circ \text{ F.} = 5^\circ \text{ C.} = 4^\circ \text{ R.}$$

Note 1.—In converting F. to C. or R. first subtract 32°, so that the number of degrees from freezing point may be ascertained; and then multiply the remainder by $\frac{5}{9}$ (for C.) or $\frac{4}{9}$ (for R.).

Note 2.—In converting C. or R. to F. add 32°, after multiplying by $\frac{9}{5}$ (for C.) or $\frac{9}{4}$ (for R.).

Examples

(1) Convert 158° F. to C.

$$158^\circ - 32^\circ = 126^\circ$$
$$\text{then } \frac{126 \times 5}{9} = \frac{630}{9} = 70^\circ \text{ C.}$$

(2) Convert 100° C. to F.

$$\frac{100 \times 9}{5} = 180$$

$$\text{then } 180 + 32 = 212^\circ \text{ F.}$$

(3) Convert 85° C. to R.

$$\frac{85 \times 4}{5} = \frac{340}{5} = 68^\circ \text{ R.}$$

CHAPTER XIX

MEASUREMENT OF WORK; HORSE-POWER OF ENGINES; CONNECTION BETWEEN HEAT AND WORK

ENERGY.—Energy may be defined as being the power of doing work, or the power of overcoming resistances of any kind. Whenever a body is put into motion, be it a solid, liquid, or gas, force or power is needed for the purpose, and the body so put into motion acquires a certain amount of energy. Thus a running stream moves by the force of gravity, and in this movement it possesses energy which may be made use of to do various kinds of mechanical work, as, for example, the turning of turbines, water-wheels, &c., which may be connected to milling or other machinery. Wind, which is air in motion, possesses this power of doing work, which is often utilized for turning windmills, moving ships, &c. When energy is thus utilized *work* is done. A horse has energy, and does work in moving tubs, wagons, or carts. Steam possesses energy derived from heat, which it exerts through the medium of an engine, and does work in moving a train of tubs, in winding cages, in pumping water, &c. Whenever force or pressure is exerted through a distance of space, and resistance is overcome, work is accomplished.

Measurement of Work.—In order that the amount of work done by the expenditure of force may be measured, it is necessary to have a standard of measurement. Standard units to express distances, weights, &c., are agreed upon, and to express and measure work a standard unit of work is adopted.

A *unit* of work is done when sufficient energy is exerted to lift a weight of 1 pound to the height of 1 foot. This amount of work is known as a *foot-pound*. Any force, therefore, that raises a pound weight to a height of 1 foot performs one unit of work, and if the same weight be raised 2, 3, or 4 feet, two, three, or four units of work respectively are done; and so on. If a 4-pound weight is raised 4 feet, sixteen units of work are done. It is therefore obvious that the amount of work done in any case may be measured by multiplying the weight in pounds by the vertical height in feet through which it is raised, and the answer will be units of work, or foot-pounds. Force, pressure, and resistance of every kind are expressed in pounds, and, in whatever direction they may be exerted, the space passed through in feet multiplied by the pounds pressure exerted gives the units of work done. A unit of work is done whenever 1 pound of pressure is exerted through the space of 1 foot, therefore it is used as the measure of every kind of work.

Horse-power.—When the weights of bodies are considerable, a higher unit than pounds is used to express them, viz. cwts. or tons; so, in like manner, when the amount of work done is considerable, it becomes inconvenient to always state it in units of work, and a higher unit is adopted, termed a *horse-power*. This unit of power was proposed and used by Watt, in order to be able to compare the power of the engines which he made with the power of horses. In most instances it was the work of horses that the engines were intended to replace, consequently it became necessary for Watt to be able to state the number of horses to which his engines were equivalent in power, or, in other words, as we now state

it, the horse-power. Watt estimated that a horse was capable of doing 33,000 units of work per minute; that is, could exert sufficient energy to raise 33,000 pounds 1 foot high in 1 minute. A horse-power, therefore, is equal to 33,000 units of work per minute, and Watt adopted this as the measure of the power of his steam-engines, and of the work they were capable of doing. It would be extremely inconvenient to express in units of work the capabilities of the large and powerful engines which are now built, consequently the term horse-power is still used for this purpose by engineers.

Examples

1. How many units of work will be expended in raising 100 tons of coals to a height of 50 fathoms?

Weight of the coals in pounds $= 100 \times 2240 = 224{,}000$.
Units of work in raising 1 lb. 50 fathoms $= 50 \times 6 = 300$.
Total units of work expended $= 224{,}000 \times 300 = 67{,}200{,}000$.

2. What horse-power will be required to lift 500 gallons of water per minute from a shaft 75 fathoms in depth?

1 gallon of water weighs 10 lb.
Weight of water to be lifted per minute $= 500 \times 10 = 5000$ lb.
Depth of shaft in feet $= 75 \times 6 = 450$.

$$\text{H.P.} = \frac{5000 \times 450}{33{,}000} = 68 \cdot 1.$$

Nominal Horse-power.—This term was proposed by Watt in order that the size of an engine might be understood by others than engineers. He assumed that all his engines would be worked by a pressure of about 7 lb. per square inch, and on this assumption calculated the nominal horse-power. When, however, steam pressures were increased much above this, the nominal and actual horse-powers, or, in other words, the calculated and the actual capabilities of the engines, differed considerably. Nowadays, when high steam pressure is generally used, the term has practically no meaning, and should be dropped.

Actual or Indicated Horse-power.—This is the measure of the capacity of an engine for doing work, or the amount of power exerted by the steam upon the pistons in the cylinders. It can be ascertained in the case of any engine when we know the mean pressure of the steam in the cylinder during the motion of each stroke; also the diameter of the cylinders and length of the stroke, and the number of strokes made per minute. The units of work done by an engine in one minute are equal to the total mean pressure in pounds of the steam on the piston, multiplied by the distance in feet travelled by the piston in one minute. Then the units of work divided by 33,000 gives the horse-power of the engine. This is expressed by the following formula:

Let P = mean pressure of steam in pounds per square inch.
A = area of cylinder in square inches.
L = length of stroke in feet.
N = number of strokes per minute.
H.P. = the horse-power.

$$\text{H.P.} = \frac{\text{P A L N}}{33{,}000}.$$

This may be more easily remembered if written $\text{H.P.} = \frac{\text{PLAN}}{33{,}000}$.

EXAMPLE.—Find the horse-power of a single-cylinder haulage engine whose cylinder is 15 inches diameter and length of stroke $2\frac{1}{2}$ feet, when the mean steam pressure is 66 lb. per square inch, and the engine runs at 40 revolutions per minute.

To complete one revolution the engine must make two strokes.

$$\therefore N = 40 \times 2 = 80.$$
$$P = 66.$$
$$A = 15 \times 15 \times \cdot 7854 = 176 \cdot 715 \text{ sq. inches.}$$
$$L = 2 \cdot 5.$$

And by formula, $\text{H.P.} = \frac{66 \times 15 \times 15 \times \cdot 7854 \times 2 \cdot 5 \times 80}{33{,}000}$

$$= 70 \cdot 68.$$

This is the work done in the cylinder and takes no account of any frictional losses; such power is described as the indicated horse-power.

The Useful or Effective Horse-power.—This term refers not to the power of the engine to do work, but to the actual work done. The quantity of water pumped, the weight lifted in winding, the numbers of tons drawn along a railway by engines, represent the useful or effective horse-power. There is a very great difference in the amount of useful work done by engines; it is a very variable quantity, depending upon their efficiency, and upon the arrangement of the machinery and of the various working parts.

MECHANICAL EQUIVALENT OF HEAT.—Having admitted that coal is the source of the energy of the steam-engine, it is necessary to ascertain how much work can be obtained by the expenditure of a given amount of heat or by the combustion of a given quantity of coal. Dr. Joule, of Manchester, conducted a series of experiments to determine this, and in 1843 arrived at the result that one unit of heat will perform 772 foot-pounds, and conversely 772 foot-pounds of work are necessary to produce a unit of heat. This is termed " Joule's Mechanical Equivalent of Heat ", and to express it shortly we say, 1 British thermal unit = 772 foot-pounds. When the calorific power of coal or other combustible used is known, the units of work may be found by multiplying this by 772. Thus, 1 lb. of coal of good quality will yield about 14,000 units of heat = $14{,}000 \times 772 = 10{,}808{,}000$ units of work. This is equal to a consumption of about $\frac{1}{5}$ lb. of coal per indicated horse-power per hour. The best constructed steam-engines require about $2\frac{1}{2}$ lb. of coal per horse-power per hour, showing that there is a loss in practice of over 90 per cent of the energy stored up in coal.

$$14{,}000 \times 2{\cdot}5 \times 772 = 27{,}020{,}000 \text{ units of work.}$$

But 1 h.p. per hour is equal to only $33{,}000 \times 60 = 1{,}980{,}000$.

$$\therefore \frac{1{,}980{,}000 \times 100}{27{,}020{,}000} = 7{\cdot}3 \text{ per cent efficiency.}$$

CHAPTER XX

PRINCIPLE UNDERLYING THE ACTION OF THE BAROMETER AND COMMON PUMP

Weight of the Atmosphere.—The fact that air has weight has been known a long time. Like all other forms of matter, air is acted on by gravity, and it exerts a pressure on the surface of the earth and on all bodies at the rate of about 15 lb. for every square inch. Human beings are not conscious, under ordinary circumstances, of this pressure upon them, because it acts equally in every direction in accordance with the general laws of fluid pressure. If, however, the pressure is reduced or increased upon one side of any person, and the equilibrium destroyed, he is immediately made sensible of the change. That air has weight and does exert a pressure upon the surface of the earth may be proved by many simple experiments. One experiment may be described. Take a glass cylinder open at both ends, place one end on the plate of an air-pump, and securely cover the other end with a piece of bladder. If the air be now pumped out of the cylinder, the bladder will burst with a report, due to the pressure of the air above acting upon it. An example of the pressure of air familiar to most miners may be given. In nearly all mines some difficulty is experienced in opening doors between intakes and returns, particularly when near the bottom of downcast and upcast shafts; this difficulty

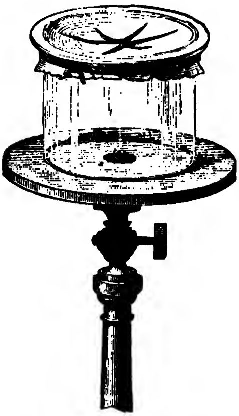


Fig. 136.—Burst Bladder—Pressure of Air Experiment

arises from the pressure of the downcast air upon the doors.

Discovery of the Principle upon which the Barometer is Constructed.—Galileo was the first who discovered that air is subject to the great law of gravitation, and has weight. He experimented to prove his assertions, and used a copper vessel into which air was condensed, the vessel increasing in weight in proportion to the quantity of air it contained, showing that the air had weight. The common lift-pump was in use in the days of Galileo to raise water from wells. As now, it would not act to a height of more than 33 feet, and the reason given was that nature abhorred a vacuum to 33 feet in height, but not higher. Galileo, however, asserted that it was not because nature abhorred a vacuum to 33 feet that water rose in a pump to that height, but because of the pressure of the air.

Torricelli, a pupil of Galileo, followed up these researches, and in 1643 demonstrated that the atmosphere has weight and exerts a pressure upon the surface of the earth, and he also discovered how to measure the pressure. He said that the reason why water did not rise to a greater height than 33 feet was because at that height the weight of the column of water counterbalanced the weight of the atmosphere. In experimenting to prove this, he constructed the first barometer. Mercury being 13·6 times heavier than water, he considered that a column of mercury of about 30 inches or $\frac{1}{13 \cdot 6}$ of the length of the column of water, 33 feet, which counterbalanced the atmosphere, would be of equal weight, and therefore also counterbalance the atmosphere. He took a glass tube about 36 inches long, closed at one end, and filled it with mercury; he then put his finger over the open end, and, inverting the tube, placed the open end in a vessel containing mercury. Having withdrawn his finger from the tube, the mercury which he had poured into it, measuring 36 inches, fell until it reached

30 inches, at which it remained, the column of mercury at that height being of equal weight to that of the atmosphere. Torricelli's theory was thus proved to be correct by the mercury standing at a height corresponding to the height of the water in the pump in the ratio of their specific gravities. A 30-inch column of mercury will balance a 33-feet column of water of the same sectional area, and each will balance the atmospheric pressure.

Atmospheric Pressure subject to Variations.—The correctness of the atmospheric pressure theory was further tested by Torricelli ascending a mountain with his crude barometer. As he went up, he observed that the mercury gradually sank in the tube, and the column became shorter. The column of air above becomes shorter in ascending; it therefore has less weight and exerts a reduced pressure. This was proved by Torricelli in ascending the mountain; the length of the column of mercury necessary to balance the atmospheric pressure gradually became less and less as he went up.

The opposite is observed when a barometer is taken into a mine. The increased column of air due to the depth of the mine is indicated by a longer column of mercury. The readings of the barometer may therefore become an approximate measure of the heights of hills above sea-level, and of the depth of mines below the surface or sea-level. For the latter, roughly, the mercury rises 1 inch for every 150 fathoms in depth.

The atmospheric pressure not only varies with altitude, but is found to undergo considerable variation even at the same place. This variation is ascertained by the rising and falling of the mercurial column in the barometer, and is measured and expressed, not by pounds and ounces, but by the length of the column in inches. The variation in England is, generally speaking, from 28 inches to 31 inches. The pressure in pounds due to a column of mercury may be ascertained by multiplying the length of the column in inches by ·4908. The standard atmospheric pressure at 32° F. and at sea-

level = 29·922 inches of mercury = 14·696 lb. per square inch = 2116 lb. per square foot, and = 33·9 feet of water column.

Principle of the Action of the Common Pump.—The action of the common pump depends, as we have already shown, upon the pressure of the atmosphere. If a vacuum be obtained in a pipe placed amongst water, the water will rise in the pipe to such a height as will counterbalance the atmospheric pressure. This height is usually about 33 feet, but varies somewhat with the variations of the atmospheric pressure. It is reduced when at an altitude, and increased in a deep mine.

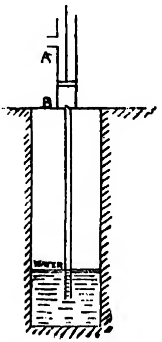


Fig. 137.—Common Hand-pump

The common pump consists of a cylindrical barrel, A B in fig. 137, in which is a bucket capable of being moved up and down by means of a rod. At the bottom of the barrel is attached a pipe, called the suction-pipe, the bottom end of which is open and immersed in the water to be pumped. The bucket is provided with a valve opening upwards, and at B is another valve also opening upwards.

Suppose the bucket to be at B; let it be raised from B to A; the air in the pipe will expand by virtue of its elasticity, and press open the valve at B and fill up the space B A of the barrel. The pressure of the air in the pipe will now be less, and will therefore exert less pressure upon the surface of the water within the pipe. Then the pressure of the outside atmosphere acting upon the water in the well will force the water up the pipe to such a height as will restore equilibrium. Let the bucket now descend; the valve at B will close, and the air in A B will escape through the bucket valve. If the bucket makes another upstroke, the above action will be repeated, and the water rise higher than before. In like

manner, if the up-and-down action of the bucket be continued, the water will continue to rise higher in the pipe, until it opens the valve at B and fills the barrel, that is, if the height from the surface of the water to A be a little less than will counterbalance the atmospheric pressure. Upon the next descent of the bucket, the valve at B will be closed, and the bucket valve will be forced open, so that the water in the barrel will be above the bucket. Then when the bucket makes its next upstroke, the water will be raised along with it and flow out of the spout A; and if the up-and-down action of the bucket be continued, there will be a continuous flow of water. There is an alternate opening and shutting of the valves at each stroke of the bucket. During the upstroke, the bucket valve is closed and the valve at B open; during the downstroke, the bucket valve is open and the valve at B closed.

From the above, it is therefore evident that if the length of the column from the surface of the water to A be greater than the column whose weight is equal to the atmospheric pressure, the pump will not act. Theoretically, pumps ought to act to a height of from 32 to 34 feet, but, in practice, owing to leakages, the height attained is generally very much shorter than this.

CHAPTER XXI

THE VENTILATION OF COAL MINES

General Proportion of Air and Gases found in Mines

The Atmosphere.—The earth is surrounded by a gaseous mixture called the *atmosphere*, or, more commonly, the *air*. It reaches a height above the surface of the earth of about 45 miles, and may be likened to an ocean of air, at the bottom of which we live.

Composition of the Atmosphere.—Air is a mechanical mixture, and not a chemical compound, as was formerly thought. It consists of oxygen and nitrogen, with small proportions of carbon dioxide, watery vapour, ammonia, argon, helium, and other rare gases. When free from carbon dioxide and moisture, its composition is approximately as follows:

		Weight.	Volume.
AIR	Oxygen ..	23	21
	Nitrogen ..	77	79
		100	100

Analyses of the atmosphere show its composition to vary very little at different points of the earth's surface. The following is the composition of pure dry air before breathing, and the composition after breathing:

		Pure Air. Volume.	Exhaled Air. Volume.
AIR	Oxygen	20·867	17·400
	Nitrogen	79·100	79·100
	Carbon Dioxide ..	0·033	3·500
		100·000	100·000

Thus about 3½ parts of oxygen are used during the process of breathing, and carbon dioxide is thereby increased about 100 times.

Properties of Air.—Air, in common with other gases, is impenetrable, inert, elastic, and ponderable. The weight of air is proved in several ways: by the barometer, the action of the common pump, and the effect upon persons who ascend to any considerable height above the earth. Air increases in weight as we descend, and decreases as we ascend; thus it weighs more on the surface of the earth or at sea-level than on places of elevation, and still more at the bottom of mines, owing to the pressure of the greater quantity of air above. The standard atmospheric pressure at sea-level, and with a temperature of 32° F., is 29·922 inches

of mercurial column, which equals 14·696 lb. per square inch. The actual weight of the air at any point usually varies from day to day, as indicated by the barometer.

The following formula is used to ascertain the weight of a given volume of air or gas:

W = Weight in pounds of a cubic foot of air.
B = Barometer: height in inches and decimals.
T = Thermometer: temperature in degrees Fahrenheit.
1·3253 = Weight in pounds of 459 cubic feet of air at 0° F. and 1 inch mercury.
459 = Cubic feet of air, which expand exactly 1 cubic foot for every increase of 1° F. in temperature.

$$W = \frac{1{\cdot}3253 \times B}{459 + T}.$$

EXAMPLE.—Find the weight of 1000 cubic feet of air at a temperature of 41° F. and a barometric pressure of 30 inches.

$$W = \frac{1{\cdot}3253 \times 30}{459 + 41} = \frac{39{\cdot}759}{500};$$

but 1000 cubic feet = W × 1000,

$$\therefore \frac{39{\cdot}759 \times 1000}{500} \quad 79{\cdot}518 \text{ lb.}$$

To find the weight of any kind of gas at the same temperature and pressure, multiply the weight of air by the specific gravity of the gas.

Combustion.—What is termed combustion is the union of oxygen with other substances, accompanied by the generation of heat and light. Oxygen has an affinity for combination with other chemical elements, especially with carbon, and as it (oxygen) exists in the atmosphere in large quantities combustion is readily set up. Thus when a light is applied to some brushwood, rapid combination of oxygen with the carbon of the brushwood is set up, with the generation of heat, and the process is termed *combustion*. The result of the combination of oxygen with carbon is the production of the gas—*carbonic acid gas* or *carbon dioxide* (CO_2), of which we shall read more presently.

Air as a Supporter of Life and Combustion.—Air is the supporter of all life and combustion. All animal and vegetable life depend for their vitality upon a constant supply of it.

The *oxygen* gas in the air is the constituent that really supports life, which is just a form of combustion, the other constituent, nitrogen, acting only as a diluent of the oxygen. Oxygen in its pure state supports violent and rapid combustion, and is quite unfit for breathing. Many substances which refuse to burn in air consume rapidly in oxygen.

Nitrogen gas is incombustible, and will not support life or lights, its properties being in this respect the very opposite of those of oxygen. When the two are brought together and mixed mechanically, as in air, they form a suitable mixture for respiration and ordinary combustion.

Quantity of Air required in Mines.—This is very difficult to determine accurately, as we should know the exact quantity required per minute by each man, horse, and light, and also the quantity necessary to dilute, render harmless, and carry away all the noxious gases generated in the mine.

It has been ascertained that an average man requires about 500 cubic inches of air per minute when at rest, and 1500 cubic inches when exerting himself in walking. Labour in a coal-mine is more violent exercise than ordinary walking, therefore the quantity required per man is increased to 1728 cubic inches or 1 cubic foot per minute. A horse requires about six times that quantity, and an ordinary safety lamp about one half.

The amount of gas evolved in a mine varies so largely, that it is impossible to calculate the quantity of air required to keep the mine in a safe condition. It has been found in practice that 100 cubic feet of air per minute per person employed in mines which are free from gases is sufficient for all purposes. The quantity in mines in which gases are given off in moderate quantities is 250

cubic feet per minute, and in very fiery mines from 300 to 500 cubic feet per minute per person employed. Although this basis of ventilation is suggested by the Mines Act, it is obvious that the real criterion of what is required depends upon the amount and character of the various noxious gases found in the return air current.

Vitiation of Air in Mines.—Air is vitiated or rendered impure in mine workings by the breathing of men and horses, the combustion of lights, the use of explosives, the various gases given off from the strata, watery vapour, coal dust, and underground or gob-fires.

The chief vitiating agents are the gases given off, and it is necessary that all persons engaged in mining should have some knowledge of their properties, and understand the effect they have, when mixed with air, upon lights and upon persons who breathe the mixture.

Gases Found in Mines

The following are the chief compound gases found in mines:

Scientific Name.	Chemical Formula.	Mining Name.	Density. Hydrogen = 1
1. Carburetted Hydrogen or Methyl Hydride ..	CH_4	Marsh-gas, firedamp, or simply gas ..	8
2. Carbon Dioxide or Carbonic Acid Gas ..	CO_2	Choke damp, black damp, stythe.. ..	22
3. Carbonic Oxide or Carbon Monoxide.. ..	CO	White damp, sweat damp	14
4. Sulphuretted Hydrogen or Hydrogen Sulphide	SH_2	Stink damp	17

Carburetted Hydrogen or Methyl Hydride.—This gas is the most abundant and the most dangerous of the gases generated in mines. It is composed of 1 volume of carbon united with 4 volumes of hydrogen, and is represented by the formula CH_4. It is a very light gas, very little more than half as heavy as air, its density being 8; taking the specific gravity of air to be 1·000, that of carburetted hydrogen is ·559.

Occurrence in Nature.—This gas is produced naturally, and may sometimes be found in marshy ground, hence

the name *marsh-gas*. It is supposed to be evolved in its pure state during the decomposition of vegetable matter, and in the conversion of vegetable matter to coal in the presence of water. Thus it is found as a natural product pent up in coal-seams and the adjacent strata, and is given off in greater or less quantities in the removal of coal from its bed. In seams which exist at a moderate depth, or are overlaid by pervious or porous strata, the gas may have escaped in a great measure, in which case they are termed *non-fiery* seams, and may be worked with exposed lights.

How given off in Mines.—It is given off from the face and sides of excavations in the coal, from the roof and floor, and from falls and faults. It is often discharged in large quantities from the latter, and from *breakers* or fissures in the roof and floor. It is generally given off in a regular manner, exuding silently from the pores of the coal, but sometimes in large quantities, at *blowers* or sudden outbursts from the coal, roof, or floor, with a noise resembling the blowing off of steam from a boiler. These blowers may be exhausted in a short time, but in some cases they continue to pour forth gas for years. Experiments have shown that gas is sometimes pent up in the pores of the coal at an enormous pressure, in some cases between 400 and 500 lb. per square inch.

Properties of Carburetted Hydrogen.—It is a colourless, tasteless, and inodorous gas. When in a pure state it causes speedy death by producing asphyxia. It is combustible, burning with a blue flame, but will not support combustion. In itself it cannot explode, as there is no oxygen in its composition to produce combustion and explosion, but when mixed with certain proportions of air it will explode. Mixed with about 5 volumes of air it burns or explodes feebly. The explosive force is increased by adding more air to it, and when the mixture reaches the proportions of 9·5 volumes of air to 1 volume of gas the maximum explosive force is reached, there being the exact quantity of oxygen in the air to com-

pletely consume the constituents of the gas. By adding more air the explosive force is reduced; when mixed with 13 volumes of air it only slightly explodes, and with more air no explosion will take place.

How Detected.—Owing to its lightness this gas when given off is usually found in the highest parts of the mine. It floats near the roof, and lodges in all cavities, breakers, and places where falls of stone have occurred. It accumulates in the goafs or gobs, which are difficult to ventilate, and may expand and show itself at the goaf edge or in the broken workings upon a decrease of atmospheric pressure. The amount of firedamp mixed with air that may be detected by an ordinary safety lamp is from 2 to 3 per cent, according to the sensitiveness of the lamp and the skill of the observer. When making the test, the flame of the safety lamp must be drawn down until there is only a faint line of blue; then if gas be present a " cap " of flame will appear above the blue line, and a practical observer is able to tell the percentage according to the size of the cap. As the percentage of firedamp increases, the blue cap increases in length and width until, with 7 per cent, the cap will fill the lamp. When the percentage exceeds 8 up to 14 the mixture explodes in the lamp, and will probably extinguish the flame of the lamp.

The detection of 2 to 3 per cent of firedamp with safety lamps is only possible where the examination is made by an experienced person, whose eyesight is suitable for testing and quickly perceiving the thin light-blue cap. Since it has become demonstrated that even less than 1 per cent of firedamp may be dangerous in an atmosphere charged with fine coal dust, the importance of being able to detect smaller percentages than is possible by the ordinary means has been realized.

Carbon Dioxide or Carbonic Acid Gas.

Its Composition.—It is composed of one atom of carbon chemically combined with two atoms of oxygen. It is represented by the formula CO_2, and is a very heavy

gas, its density being 22. The specific gravity of air being taken as 1·000, that of carbon dioxide is 1·529.

Occurrence in Nature.—It is found in small proportions in air and water. It is given off by volcanoes and in ordinary combustion. It is given off in mines from the strata, explosion of blasting agents, combustion of lights, and by the breathing of men and horses. It forms one of the constituents of the gas termed *after-damp*, which results from an explosion of firedamp.

Its Properties.—A colourless gas, with a slightly acid taste and smell. It acts as a narcotic poison, and is very dangerous to animal life. It will not support combustion, and will not burn, but immediately extinguishes lights. Air containing about 4 per cent of it will extinguish lights, and is dangerous to breathe. It is sometimes produced artificially in mines to extinguish gob-fires, and may be prepared in large quantities by decomposing calcic carbonate with hydrochloric acid.

How given off in Mines.—It is found in all mines more or less, being given off by the respiration of the persons and animals employed. The coal and adjacent rocks give it off in a noiseless manner, more of it being found in shallow damp mines than in deep dry ones. In shallow mines the quantity given off is affected by the direction of the surface winds and the atmospheric pressure. Return air-ways usually contain a large proportion of this gas and watery vapour. Timber in these air-ways rapidly rots and decays owing to the presence of this gas. Being of great density, it is often found in low places, on the floor of the seam, in dip workings, in old unventilated workings, and at the bottom of unventilated shafts and wells.

How Detected.—The presence of this gas is detected by the light of the lamp being dimmed or extinguished according to the percentage of it in the air. It produces a sleepy feeling in persons breathing it, and affects their limbs. If they are overcome and fall to the floor they soon perish, as it lies more deadly upon the floor owing

to its great density. If necessary at any time to descend an unused shaft or well a light should first be let down, or the person descending should be secured and carry a light, and as soon as the latter becomes dimmed he should return to the surface until the gas which he has detected has been cleared out.

Carbonic Oxide or Carbon Monoxide.

Its Composition.—This gas is composed of one atom of carbon chemically combined with one atom of oxygen. Its density is 14, and it is represented by the formula CO. Taking air at 1·000 the specific gravity of this gas is ·967—nearly the same as air.

Its Occurrence.—This gas results from imperfect or incomplete combustion. When there is sufficient air carbonic acid gas is the result of combustion, but if there is a deficiency and the combustion of the carbon is incomplete, carbonic oxide is produced. It is produced in mines when incomplete combustion occurs at an explosion of some blasting agent, at gob-fires, and after an explosion of firedamp.

At Crarae Quarry, in Argyllshire, on 25th September, 1886, seven persons were killed and many injured by the inhalation of poisonous gases shortly after the explosion of a heavy charge of gunpowder. The chief gas present was supposed to have been carbonic oxide.

Its Properties.—It has neither colour, taste, nor smell. It is slightly soluble in water, and is an exceedingly poisonous gas. It will itself burn, but does not support combustion. A very slight percentage of it in air produces giddiness and fainting when breathed, and 1 per cent causes death.

How Detected.—It is said that a small amount of it in air causes lights to burn brighter, but its action upon them is not sufficient to serve for detecting its presence. When present in air it may be detected by its effect upon the persons breathing it. In minute proportions it causes trembling in the limbs, loss of strength, and severe headache. This gas only occurs in mines as the

result of exceptional circumstances, as, for instance, a gob-fire. When suspected, its presence may be detected by its action on a small warm-blooded animal. Birds, such as linnets or canaries, or mice are taken into the suspected atmosphere, and as they are affected in much less time than a man, an indication of the dangerous nature of the atmosphere is obtained in time to give the men entering it warning.

Sulphuretted Hydrogen or Hydrogen Sulphide.

Its Composition.—It is composed of two atoms of hydrogen chemically united with one atom of sulphur. It is represented by the formula SH_2, and its density is 17. Its specific gravity is 1·171 as compared with air 1·000.

Its Occurrence.—It is produced by the decay of vegetable and organic matter, in the presence of certain compounds of sulphur, and is found in waters which contain organic matter and sulphates. It is sometimes found in the water which is found in mines.

Its Properties.—It is transparent, colourless, and has a disagreeable odour like that arising from rotten eggs, a property which enables it to be easily recognized. It is a powerful narcotic, and if inhaled, even when largely diluted with air, acts very injuriously upon the system. Authorities are not agreed as to the exact proportion of this gas in air which is fatal to human life; 1 per cent is said to produce fainting fits. It does not support combustion, but is in itself combustible—that is, it will burn in the presence of air with a blue flame. It dissolves in water, which takes up about three times its volume of it, and imparts to the water its peculiar and offensive smell and slightly acid taste. Its presence in the mine may be due to spontaneous combustion. In fact it is generally the presence of this gas that produces what is known to the miner as " fire-stink ". It is unusual to find more than mere traces of this gas in the mine, and it is fortunate that it is so, for the gas is an extremely dangerous one when inhaled even in minute quantities.

How Detected.—This gas is most easily detected by its disagreeable odour, which, as we have already said, resembles that of rotten eggs. Lights are of no service in its detection, as they continue to burn brightly in a mixture unsafe to breathe.

After-damp.—In addition to the four gases just described, there is another mixture which sometimes forms the atmosphere of mines—namely, after-damp, which is simply the gases resulting from an explosion of firedamp. It is a most deadly mixture, causing insensibility and death very quickly when breathed. The air in the mine at the moment of the explosion is deprived of its life-giving gas, oxygen, which unites with the CH_4, and the composition of the resulting mixture is:

	By Volume.
Nitrogen	71·4
Carbonic acid gas	9·6
Steam	19·0
	100·0

The steam condenses into water soon after the explosion and leaves the nitrogen and carbonic acid gas free. Both are unfit to breathe, and in many cases carry death to a greater number of persons than have been killed by the force of the explosion. Carbonic oxide is sometimes a constituent of after-damp, especially if coal dust be present at the time of the explosion. In Messrs. Atkinson's *Explosions in Coal Mines*, the following analysis is given of a sample of the after-damp taken after the Usworth Colliery Explosion in 1885:

	Volume.
Carbonic acid gas	4·54
Carbonic oxide	2·48
Light carburetted hydrogen	8·68
Oxygen	7·23
Nitrogen	76·80
	99·73

According to Dr. Haldane's observations on the causes of death in colliery explosions, it appears that death from carbon monoxide poisoning may be recognized from the fact that the colour of the blood after death is more or less red instead of blue. The bodies thus often present an extraordinary lifelike appearance. After-damp is generally the chief obstacle in the way of exploring parties anxious to penetrate the workings of a mine where an explosion has occurred. The arrangements for producing and distributing throughout the various districts of the mine currents of fresh air are generally injured or destroyed by an explosion, and for a time the ventilation is stagnant; thus the conditions are favourable for the after-damp to do its deadly work, and for a fresh accumulation of firedamp and poisonous gases, which may be given off in the mine, to further increase the dangers and difficulties of exploration.

Very great care is necessary, on the part of the persons entering a mine after an explosion, not to advance beyond the point where the air is breathable, and as sometimes the greatest damage to the stoppings, air-crossings, and doors, which direct the air around the mine, is done near the bottom of the downcast, it is consequently impossible to proceed very far until some of the damage is repaired. Under the conditions prevailing after an explosion the flame of a safety lamp is not a sufficiently reliable index of the nature of the atmosphere, as it may continue to burn where carbon monoxide is present in deadly proportions. It has therefore become customary for the explorers to have with them small live animals, such as mice or canaries, which quickly give indications of the dangerous nature of the atmosphere before any serious effects are felt by the explorers or shown by the safety lamps.

Dr. Benson, in his evidence on the immediate causes of death in the West Stanley Explosion, 17th February, 1909, stated: "(1) Those killed by carbon monoxide poisoning numbered 121; (2) those killed by direct

violence numbered 25; (3) deaths due to other causes numbered 19 ".

Occluded Gases.—Coal usually contains gases pent up in its pores, which exude when the coal is laid bare. These gases, which are termed *enclosed* or *occluded* gases, are separate, and must be distinguished from those which make up the chemical composition of coal. Gas, such as firedamp or stythe, given off at a working face of a seam of coal either quietly and slowly or in a sudden outburst, has been stored up in the pores of the coal. The quantity and the rate of discharge of gases are very variable, depending, to some extent, upon the nature and structure of the coal. It has been found from experiment that tender and porous coals yield up their occluded gases very much quicker than do the dense hard coals. This to a large extent is true in practice; porous and tender seams of coal are frequently " fiery ".

CHAPTER XXII

THE VENTILATION OF COAL MINES.—Continued

Distribution of Air through the Workings of Mines. Stoppings. Doors. Regulators. Air-crossings. Laws affecting the Circulation of Air in Mines. Splitting of Air.

What is Ventilation?—The ventilation of a mine is the keeping up of a regular supply of pure fresh air to remove and take the place of the impure air that is generated, so that the various parts of the workings of the mine may be kept in a fit condition for the persons working therein. From what we know of gases given off in mines, it will be obvious to all that their quick removal is necessary. The Act requires that " An adequate amount of ventilation shall be constantly produced

in every mine to dilute and render harmless inflammable and noxious gases to such an extent that all shafts, roads, levels, stables, and workings of the mine shall be in a fit state for working and passing therein, and in particular that the intake air-ways up to within one hundred yards of the first working place at the working face which the air enters shall be normally kept free from inflammable gas ".

Distribution of Air through the Mine.—Air in regular, steady currents, called *ventilating currents*, being

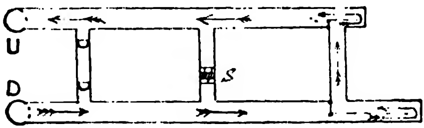


Fig. 138.—Illustrating the Conducting of Air through Workings
D D, Wood doors S, Brick stopping. Arrows indicate air current.
- - - Brattice.

required in every mine, it is necessary to provide some force or power to produce them. Winds or air currents on the surface are the result of a difference in the pressure at different points, the greater pressure forcing the air towards the point of lesser pressure in order to establish equilibrium. Air currents passing through mines are caused in a similar manner; the density of the air in one shaft is reduced by natural or artificial means, to be presently described, and the heavier air in the other shaft descends and moves in the direction of the shaft in which the air is rarefied. In doing so it is conducted around the workings of the mine, and as it passes up the shaft it, in its turn, is rarefied, and so keeps up a continuous current of air.

The shaft in which the fresh air passes from the surface into the mine is termed the *downcast*, and the shaft in which the impure air from the mine returns to the surface is termed the *upcast*.

When two shafts are sunk, and there is only one communication between them, the ventilation is most simple, as it passes down one shaft, through the passage, and up the other shaft. The work of guiding and distributing the air commences, however, as soon as more communications are made by driving away the main levels, roads, and winning-places which require to be ventilated. Fig. 138 shows a pair of drifts, one being driven from the bottom of the downcast and the other in the same direction from the bottom of the upcast, and they are connected at intervals by a communication termed a *holing* or *stenton*, at right angles to the drifts.

Stoppings.—In order to force the air current to the face of the drifts, it is necessary to barricade or " stopping off " the stentons, except the one nearest the face, otherwise the greater part of the air would take the shortest route to the upcast, which would be through the first stenton. Stoppings between main air-ways are built with brick or stone and lime with 3 yards of packing, and are termed *permanent* stoppings; if built of wood to keep the air on the face headways, they are termed *temporary* stoppings. They are built from floor to roof, and from side to side of the excavation, and are for the purpose of directing the air along the desired course.

The roadways along which the air travels from the downcast shaft to the face of the workings are the *intakes*, and those along which it travels back to the upcast shaft after sweeping around the working faces and taking up the noxious gases are termed the *returns*.

Doors.—In many instances it is necessary to pass from the intake into the return air passages at various points between the shafts and the face; where this is required, part of the stopping is removed and a wood door fixed in the masonry, opening towards the intake. Two doors are generally fixed, one a short distance away from the other, so that when one door is open for men,

horses, or tubs to pass through, the other is kept shut to prevent a sudden rush of air from the intake into the return. Doors which are intended for men only to pass through are termed *manhole doors*, and are made from 2 feet to 3 feet square, and should be securely locked. Those on main roads are made large enough for the passing of horses and tubs. In all cases the doors and framework should be set a little out of the perpendicular, so as to cause them to fall and shut by their own weight, and they should be so constructed as to prevent much leakage of air.

Near the working face, when doors are required, canvas or brattice cloth, either single or double, is used instead of wood, as the air currents are not strong. They are nailed at the top to a piece of timber, and hang down to the floor of the air-way.

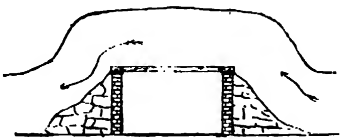


Fig. 139.—Wooden-topped Air-crossing

Air - crossings.—In the many ramifications of the workings of mines, it frequently occurs that the intake and return currents must cross each other. It is essential that the two currents should not mix, therefore one must be taken over the other by means of an air-tight bridge, termed an *air-crossing*. Usually the return current is taken over the intake, in which case the air-crossing is termed an *overcast*; sometimes it is taken under the intake, and is termed an *undercast*.

A form of air-crossing much used is the wooden-topped one (see fig. 139). It consists of two upright walls built in the return air-way parallel to the sides of the intake, and up to the height of the roof of the latter. Where the air-crossing is intended to be made, roof stone must be taken down to give sufficient height for the return air-way over the top. On the top of the side walls is placed a balk of timber, and then pieces of timber are

stretched from side wall to side wall across the intake and nailed to the balks at each side. The cross pieces of wood should have the joints feathered and grooved and plastered over the top with lime, to make the air-crossing practically air-tight.

Another form of air-crossing, one which is used where strength is required, consists of an arch over the intake instead of a flat wooden top. This form of air-crossing is shown in fig. 140. Side walls of masonry are built up, and then the arch turned of the required dimensions, stone having already been taken down to form the return air-way over the top of the arch.

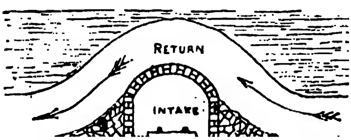


Fig. 140.—Arched Air-crossing, showing Return Air carried over Intake Air

A natural air-crossing is one made in the stone, either over or under the intake, so that the two air-ways do not come into contact. This is an expensive method, but it is the only air-crossing that seems likely to survive the force of an explosion of firedamp. In most explosions the artificial air-crossings are nearly all partially or totally destroyed.

Splitting the Air.—Formerly mines were ventilated by one continuous air current, which travelled from the downcast around the face of all the working places, and returned to the upcast usually heavily laden with gases, and in a more or less inflammable condition. There was only one intake in the mine, along which the total quantity of air entering travelled to the face. A very large number of doors were required on the main roads between the intake and return air to communicate with the various working faces of the mine, and, as a consequence, the ventilation was frequently deranged, and dreadful explosions often occurred.

The approved method of ventilation is to have separate intakes for the various districts of the mine, commencing

as near to the bottom of the downcast as possible. The total volume of air coming down the shaft splits or divides itself into portions where the main roads or intakes branch off, and each portion travels along its own separate intake to the face of the workings of the district, and returns along a separate air-way, to be reunited to the other splits at some point near the bottom of the upcast. Thus, supposing there are two main roads from the bottom of the downcast, one going straight west and the other straight east, the air on reaching the shaft bottom will split, one portion going west and the other portion east. The west split may be again split into, say, four separate splits at a short distance away from the shaft to ventilate four districts, and the same may be done on the east side.

The quantity apportioned to each district depends upon the total quantity of air and the number of splits. The total quantity of air circulating should be large enough to give to each district sufficient for its requirements. The quantity required varies according to the extent of the district, the number of men employed, and the amount of gas given off.

In most mines the quantities passing into the various splits must be *regulated*, otherwise the districts a short distance from the shafts would take too much air, and very little would be left for the districts a long distance away. This is done by fixing in each return air-way where required a slide door, termed a *regulator*, by means of which the size of the air-passage can be reduced to such dimensions as will allow of only the required quantity passing through.

Some of the advantages derived from splitting are:

1. Each district is supplied with a separate current of air, which passes along the main roads of the mine, upon which are the persons and animals engaged in the haulage of the coal.

2. A much larger quantity of air passes into the mine than when there is no splitting.

3. The atmosphere in every part of the mine is purer and safer.

4. The return currents are not so highly charged with gas, and the chances of accumulations of gas in the workings are less than when there is one long current.

5. The velocity of the different air-currents is much reduced.

Splitting the air should be judiciously arranged. The following rules may serve as a guidance:

1. The various splits should be made as near the bottom of the downcast as possible.

2. Each of the main splits should have a separate intake and return, and each of the return splits reunite close to the bottom of the upcast.

3. Where possible the air-ways and districts should be so arranged that regulators will be unnecessary.

4. Splitting should not be done too often, otherwise the velocity of the currents will be insufficient to forcibly remove gases.

The limit to which splitting may be judiciously carried is reached when the velocity of the air at the face is reduced below 2 feet per second, but much depends upon the area of cross-section of the air-way: the larger the cross-section the lower the effective velocity to pass the required quantity.

Laws affecting the Circulation of Air in Mines.

Friction of Air.—Air possesses the properties of elasticity and compressibility, and has an average weight or pressure on the surface of the earth of 14·7 lb. per square inch. Air in motion, or wind, is caused by a difference of this pressure between two or more places; the air at the place of greater pressure moves towards that of the lesser pressure. A ventilative current or wind in a mine is produced by the air in one of the shafts being rarefied or reduced in pressure, and the greater pressure of the air in the other shaft causes the air to move towards the

shaft where the pressure has been reduced. The difference of the pressure between the air in the two shafts is termed the ventilating pressure.

The speed or velocity at which the air will flow towards the place of reduced pressure depends upon the difference of pressure. The theoretical velocity of air in rushing towards a vacuum is the same as the velocity that a falling body would attain at the bottom of such a column of air (of the density of the flowing air) as, by its weight, would produce the same pressure as that which gives rise to the wind.

$$v = \sqrt{2gh}.$$

v = velocity in feet per second.
g = gravity or 32·2.
h = height from which a body must fall to produce this velocity.

$$\text{Or, } v = 8\sqrt{h}.$$

The length of a column of air representing the difference of pressure or weight of the air in the upcast and downcast shafts of a mine is termed the *motive column.*

Wind in rushing from one place to another encounters resistances, such as the surface of the land, houses, trees, and other large objects. These produce friction, which tends to retard the progress of the wind and reduce its velocity.

The ventilating currents in mines are confined to long, narrow passages or air-ways, and, in flowing along, the air meets with resistance from the surfaces of the air-ways, timber, stones, and other obstacles. The friction resulting is found to be considerable, the greatest amount being due to the air rubbing against the roof, sides, and floor of the air-ways. A very much greater proportion of the difference of pressure of the air in the upcast and downcast shafts producing the necessary ventilation in the mine is employed in overcoming this friction than in producing the velocity of the air-current. This may be shown by the water-gauge, the difference between the

actual water-gauge at the mine and the theoretical water-gauge to produce the velocity at which the air travels is the measurement of the force required to overcome the friction. The number of square feet of surface rubbed against by the air-currents of mines is termed the rubbing surface, and is ascertained by adding together the height of each side and the width of the top and bottom; the sum being multiplied by the length of the air-way equals the amount of rubbing surface.

EXAMPLE.—An air-way is 6 feet square and 1000 yards long. What is its perimeter, sectional area, and rubbing surface? What will be the velocity of a current of 21,600 cubic feet of air per minute passing through the air-way?

Perimeter = $6 + 6 + 6 + 6 = 24$ feet.
Sectional area = $6 \times 6 = 36$ feet.
Length = $1000 \times 3 = 3000$ feet.
Rubbing surface = perimeter × length = $3000 \times 24 = 72{,}000$ sq. feet.
Velocity = quantity of air ÷ area = $21{,}600 \div 36 = 600$ feet per minute.

Three Laws of Friction.—The following three laws of friction have been given and thoroughly explained by the late Mr. J. J. Atkinson in the *Principles of Ventilation.*

1. The pressure required to overcome the friction of air increases and decreases in exactly the same proportion that the area or extent of the rubbing surface, exposed to the air, increases or decreases, so that when the velocity of the air and the sectional area of the air-way remain the same, the pressure required to overcome the friction is proportioned to the area and extent of the rubbing surface exposed to it; and hence, if we double or treble the extent of the rubbing surface, we also double or treble the friction, or, what is the same, the force or pressure required to overcome it.

2. The pressure required to overcome the friction of the air increases and decreases inversely as the sectional area of the air-way increases and decreases.

This means that, if there are two air-ways, one of which is exactly double the area of the other, and the velocity of the air and the extent of the rubbing surface are the same in each, only one-half the pressure applied to each square foot of the larger air-way as applied to the smaller air-way will be needed to overcome the same amount of friction.

3. The pressure required to overcome the friction of the air increases and decreases in the same proportion that the velocity squared of the air increases and decreases; in other words, the friction varies directly as the square of the velocity.

A little consideration will serve to make this law plain. When the velocity is doubled, the air meets with $2^2 =$ a double-double or four-fold resistance. Double the quantity of air passes through the air-way in a given time when the velocity is doubled, and every part of this double quantity meets the resistance with a double velocity. If the velocity be trebled, the resistance is increased $3^2 =$ a nine-fold resistance, and so on. In the same way a reduction in velocity of $\frac{1}{2}$, $\frac{1}{3}$, $\frac{1}{4}$, &c., decreases the resistance in the proportions of $\frac{1}{2} \times \frac{1}{2} = \frac{1}{4}$; $\frac{1}{3} \times \frac{1}{3} = \frac{1}{9}$; and $\frac{1}{4} \times \frac{1}{4} = \frac{1}{16}$.

From the laws given above, the following results are obtained, by which the problems arising in connection with the ventilation of mines may be worked:

1. The pressure required to force the air through air-ways varies directly as the extent of the rubbing surface.
2. The pressure required is inversely proportional to the area, other conditions remaining the same.
3. The pressure varies directly as the square of the velocity.
4. The pressure varies directly as the length of the air-way.
5. The quantity of air circulating varies as the square root of the pressure.
6. The quantity of air passing in air-ways of different areas, other things being equal, is according to the square root of the area multiplied by the area.
7. The quantity varies as the cube root of the power.
8. The quantity varies inversely as the square root of the rubbing surface.

Let p = the pressure of the air in pounds per square foot producing ventilation.

a = the sectional area of the air-way in square feet.

s = the extent of rubbing surface in square feet exposed to the air.

v = the velocity of the air in feet per minute.

k = the coefficient of friction, a constant number, found by experiment. It is equal to the ventilating pressure required to overcome the resistance that a unit of air flying with unit velocity would meet with in circulating round a mine of unit area, and having unit rubbing surface. The exact value of the coefficient of friction for the rough surfaces of the air-ways of mines varies with the nature of the surfaces, but the average is usually taken as ·01 lb. per square foot of rubbing surface per thousand feet velocity per minute.

v = velocity of air in thousands of feet per minute $= \dfrac{v}{1000}$.

Q = quantity of air passing in cubic feet per minute.

WG = water-gauge in inches $= \dfrac{p}{5{\cdot}2}$.

HP = horse-power producing ventilation.

Stated algebraically p varies as s,

$$p \quad ,, \quad v^2,$$

$$p \quad ,, \quad \frac{1}{a}.$$

$$\therefore p \text{ varies as } \frac{sv^2}{a};$$

but as the real value of p depends on the coefficient of friction,

$$p = \frac{ksv^2}{a}.$$

EXAMPLE.—An air-way has a cross-section 12 feet by 6 feet, and is 1000 fathoms long. Find the pressure required to pass a ventilating current of 10,000 cubic feet of air per minute through it.

Length = 1000 × 6 = 6000 feet.
Area = 12 × 6 = 72 sq. feet.
Perimeter = 12 + 6 + 12 + 6 = 36 feet.
Rubbing surface = perimeter × length = 36 × 1000 × 6 = 216,000 sq. feet.

Velocity in feet per minute

$$= \frac{\text{quantity in cu. ft. per min.}}{\text{Area}} = \frac{10{,}000}{72} = 138\tfrac{8}{9}.$$

Velocity in thousands of feet per minute

$$= \frac{v}{1000} = \frac{139\frac{8}{9}}{1000} = \frac{5}{36}.$$

$$v^2 = \frac{5}{36} \times \frac{5}{36} = \frac{25}{1296}.$$

$$p = \frac{ksv^2}{a} = \frac{\cdot01 \times 216{,}000 \times 25}{72 \times 1296} = \cdot578 \text{ lb.}$$

The terms *velocity*, *pressure*, *power*, and *quantity* are much used in connection with ventilation, and it is important that they be thoroughly understood.

Velocity is the speed at which the air-currents flow along the air-ways; it is measured by an anemometer in lineal feet per minute.

Pressure is the force in pounds per square foot producing the ventilative current. Power is the units of work in the current, and is ascertained by multiplying the quantity of air by the pressure. The difference between pressure and power should be particularly noted.

Quantity is the volume of air passing through the mine in a given time. It is usually expressed in cubic feet per minute, and is found by multiplying the area of the air-way by the velocity.

To ascertain the horse-power in a current of air:

$$\text{Horse-power} = \frac{\text{c. ft. per min. of air} \times \text{water-gauge} \times 5\cdot2}{33{,}000}.$$

EXAMPLES.—Find the horse-power represented by a current of air of 120,000 cubic feet per minute with a water-gauge = 2·5 inches.

$$\text{Horse-power} = \frac{120{,}000 \times 2\cdot5 \times 5\cdot2}{33{,}000} = 47\cdot27.$$

If the horse-power producing ventilation is 33·09 and the

water-gauge reads 2·1 inches, how many cubic feet per minute of air will be circulating in the mine?

$$\text{Quantity} = \frac{33{,}000 \times \text{horse-power}}{\text{water-gauge} \times 5{\cdot}2}$$

$$= \frac{33{,}000 \times 33{\cdot}09}{2{\cdot}1 \times 5{\cdot}2} = 100{,}000.$$

The water-gauge of a mine reads 3 inches, and the quantity of air circulating per minute equals 150,000 cubic feet. Find the useful horse-power.

$$\text{Horse-power} = \frac{150{,}000 \times 3 \times 5{\cdot}2}{33{,}000} = 71.$$

It should be noted that the quantity of air in a mine does not increase in the same proportion as the power, but as the cube root of the power. To double the quantity of air the horse-power must be increased $2^3 = 2 \times 2 \times 2 = 8$ times; and to treble the quantity it must be increased $3^3 = 3 \times 3 \times 3 = 27$ times.

EXAMPLE.—If by the application of 10 horse-power a current of 50,000 cubic feet per minute is produced, how many horse-power will be required to produce 100,000 cubic feet per minute?

$$50{,}000^3 : 10 :: 100{,}000^3 : 80 \text{ horse-power}$$

EXAMPLE.—If a current of 100,000 cubic feet of air per minute is produced in a certain air-way by 80 horse-power, what horse-power will be required to produce 25,000 cubic feet of air per minute in the same air-way.

$$\frac{Q_1}{Q_2} = \frac{100{,}000}{25{,}000} = \frac{4}{1}.$$

But horse-power varies as Q^3.

$$\therefore \frac{Q_1^3}{Q_2^3} = \frac{64}{1},$$

and horse-power required to pass 25,000 $= \dfrac{80}{64} = 1{\cdot}25.$

Ventilation of the Face of Working Places, Levels, Stone Drifts, &c.—The currents of air are kept up to the " innermost holing ", or the communication nearest to the face, by means of stoppings. But as the working places advance beyond this holing or air-way, the air at the face may become impure or charged with gases, and means must be applied for their removal.

The usual way is to direct the air into the face of the working place by means of canvas or wood brattice, which divides the place into two portions, one being made the intake, the other the return.

In ordinary coal working places, such as bords and headways, canvas cloth is generally used. It is hung in lengths from timber at the roof to the floor, and kept within a few feet of the face, and made as air-tight as possible. Excavations yielding little firedamp may be driven 50 yards in advance of a holing by the use of canvas cloth properly applied.

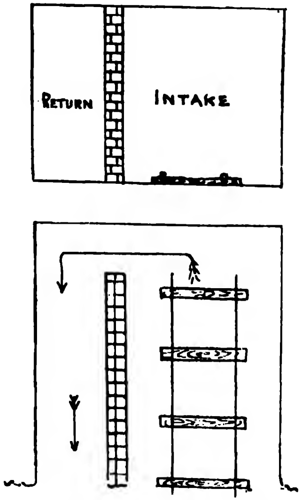


Fig. 141.—Section and Plan of Brick Brattice in Stone Drift

Wood brattice is often used where much gas is given off, necessitating a brisk air-current at the face, or when the excavation is intended to be driven singly for a long distance, as in the case of stone drifts. The wood brattice is usually about half an inch thick, and cut into suitable lengths and widths. When nailed to the upright props the joints are overlapped, and are in some cases plastered with lime to prevent leakage of the air.

In some cases, as for instance a stone drift which is to be driven a long distance, a brick brattice or wall $4\frac{1}{2}$ to 9 inches in thickness built with lime between roof and floor is adopted (see fig. 141).

Brattice is not carried up the centre of the excavation, but near to one side, making two unequal divisions. The intake air sometimes passes up the narrow side or

"behind the brattice" to the face, and returns upon the wide side where the tramway is laid (see figs. 110 and 138).

Air boxes, or pipes of wood, fire-clay, or iron, are often used to ventilate narrow places and those driven in unlevel strata, as through faults, and where the roof is very weak. Ordinary bratticing often cannot be applied in such situations; the air is therefore made to pass through boxes or pipes to the face and return along the excavation, or vice versa.

Form and Size of Air-ways.—From a theoretical point of view the circular air-way is no doubt the best, because it gives the greatest area with the least rubbing surface. Taking a circle whose area is 1 sq. yard we find the perimeter, which is the rubbing surface per unit length, 3·545 yards; whereas a square whose area is also 1 sq. yard has a perimeter equal to 4 yards per unit length. The rectangular form is worse in this respect than the square, for if we take a rectangular air-way of 1 sq. yard area, whose width is 2 yards and height ½ yard, we find the perimeter to equal 5 yards, which is about 1½ yards more than the circular and 1 yard more than the square per yard of length.

Square and rectangular air-ways are chiefly adopted in mines owing to the impracticability of the circular. The square form should, if circumstances admit, be adopted for main air-ways in preference to the rectangular.

The size of air-ways depends to some extent on the quantity of air required in the mine. It should be remembered that a small number of large air-ways is preferable to a large number of small ones for the purpose of ventilation.

CHAPTER XXIII

THE VENTILATION OF COAL MINES.—CONTINUED

NATURAL VENTILATION

What is Natural Ventilation?—As already stated, to produce a current of air in the mine a difference in the density of the air in the two shafts or outlets must be created, and the air then will move from the greater in the direction of the lesser density.

Sometimes a difference in density arises owing to the natural conditions of the mine, or to an unlevel surface between the two shafts, so that one is deeper than the other, or to one shaft being wet and the other dry. A current of air is thereby produced, and the mine is said to be ventilated by Natural Ventilation.

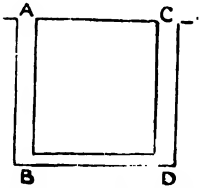


Fig. 142.—No Natural Ventilation

When there is a difference in the surface level of the two shafts or outlets, the surface temperature is the chief cause of the air current. Taking the average temperature to be about 50° F. at a depth of 90 feet, below this point the temperature increases 1° F. for about every 60 feet of descent. The shaft sides will therefore be warmer or colder than the surface according to the summer and winter season of the year.

Air in passing through a mine is increased in temperature by the heat of the strata, the heat arising from the bodies of the men, boys, and horses, and of the lights used.

In the example given in fig. 142, no natural ventilation would take place, because the two shafts, AB, CD, are level both on the surface and underground. The

shafts being of equal area and temperature, there is no natural cause to produce a current.

Fig. 143 shows a case of natural ventilation arising from a difference of surface level, assisted by the natural heat of the mine, and the heat given off by the workers. The current in summer will pass in the direction indicated by the arrows—down the shaft GH, through the drift HF, and up the shaft FE. In winter the direction will be reversed. The air-current will pass down EF,

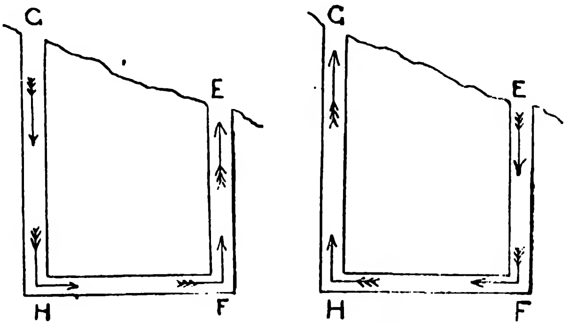


Fig. 143.—Natural Ventilation in Summer

Fig. 144.—Natural Ventilation in Winter

through FH, and up HG, as shown in fig. 144. The reason of this is found in the variation of the surface temperature. In summer the surface temperature at E will be greater and the air lighter than in the shaft GH, consequently the air in GH moves towards E. In winter the air at the point E is colder and of greater density than in the shaft GH; the air therefore reverses and moves in the opposite direction towards G.

Natural ventilation, having to depend chiefly upon such a variable source as the surface temperature, is not at all to be relied upon for the ventilation of mines. At best it produces only a small current, which may be reduced, stopped, or reversed at any moment by a

change of surface temperature or direction of wind. The reversal of the air is a most dangerous feature; noxious gases may be driven out of the returns into the working-places and intakes and produce fatal results. Natural ventilation is relied upon only at small, unimportant mines working seams near the surface.

ARTIFICIAL METHODS OF PRODUCING VENTILATION

Necessity of Artificial Ventilation.—Owing to the irregularity and insufficiency of the currents produced by natural ventilation, it is necessary to employ some artificial means to produce an adequate amount of ventilation which will continue to flow in regular quantities in one well-defined direction, viz. down the downcast shaft, through the workings, and then up the upcast shaft, without any variation.

Artificial Methods Employed.—The chief of these are: the wind-cowl, waterfall, steam-jet, furnace, and the mechanical ventilators which include the varying capacity machines and fans.

Merivale, in his *Notes and Formulæ for Mining Students*, says, " A difference of pressure may be produced by:

" A. Reducing the density of the air in the upcast shaft (depressive, exhaustive, or negative ventilation) by means of:

1. The natural heat of the mine.
2. Furnace.
3. Steam-jet.
4. Exhaust-fans.
5. Varying capacity machines.

" B. Increasing the density of the air in the downcast shaft (compressive, blowing, or positive ventilation) by means of:

1. Air-pump.
2. Waterfall.
3. Wind-cowl.
4. Blowing-fans.
5. Varying capacity machine."

The depressive system is adopted in this country, almost without exception; the compressive system is sometimes adopted in America and has been tried in Germany.

The Wind-cowl is used for temporary purposes only, and can be applied only when the surface wind is blowing. It is fixed at the top of the downcast and a pipe is carried from it down the shaft. It is turned towards the wind, the pressure of which forces the air down the pipe into the mine.

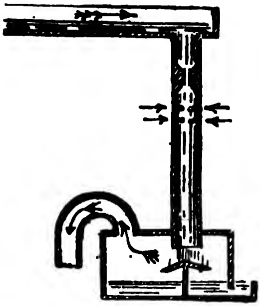


Fig. 145.—Water Trompe. Arrows indicate air-current

The Waterfall.—Small mines, such as metal mines, which have an adit level to conduct away the water, are sometimes ventilated by a waterfall. The water is scattered and made to fall down the shaft like rain, thereby producing a downward current of air.

Sometimes, instead of allowing the water to fall down the open shaft, a water trompe is adopted. This consists of a vertical pipe, contracted a little just below the top end, where the water falls into it, and immediately below the contracted part there are small holes through which the air is drawn from the outside and carried down by the water. The water falls upon a dash-block in the cistern at the bottom, and passes away at the overflow, and the air which has been separated from it passes away along the air pipes to where it is required in the mine (see fig. 145).

The waterfall is rarely adopted as a permanent means of producing ventilation, unless at a small mine where there is a good supply of surface water which will run out of the mine again by a level drift or adit. If there be no adit, and the water has to be lifted out of the mine to the surface by pumping appliances, the waterfall

will prove to be a very costly arrangement. Sometimes it is used as a temporary expedient when the ventilating machinery has been deranged, as in the case of an explosion.

Steam-jet.—The direct discharge of high-pressure steam into the upcast shaft was at one time much advocated as a means of producing ventilation, but experiments proved it to be very expensive and generally inapplicable.

The steam-jet arrangement sometimes consists of a pipe from the steam-boilers carried down the shaft some depth, and the steam is allowed to rush out of the end of the pipe. A better arrangement than this is to have the down-pipe connected to a pipe or pipes passing around the shaft, on the top of which are a number of small holes, through which the steam issues in small jets. This produces a better result than the other, because the steam acts upon a larger surface of air. The issue of steam in this way imparts momentum to the air in addition to heating it. The steam-jet is almost useless in a wet shaft, because the steam is condensed and falls down the shaft, and so impedes instead of increases the current. The steam-jet is very readily applied, and proves useful in an emergency when the ordinary ventilating arrangements have been damaged.

Furnace.—The furnace is placed a short distance away from the bottom of the upcast shaft, and produces ventilation by rarefying or reducing the density of the return air. The heat of the fire or furnace expands the air, making it lighter, so that it ascends, while the heavier air in the downcast descends to take its place.

The roof and sides of the excavation where the fire burns are usually protected by a thick brick arch resting upon side walls. If the excavation is in the solid stone, side walls are sufficient; but if it is in the seam, side coal must be removed to allow of an archway on each side of the furnace for the air to pass along to prevent ignition of the coal. This is shown in figs. 146, 147,

which illustrate the ground plan and cross-section of a furnace.

In many cases the return air of the mine is passed over the furnace and feeds the burning coal. When the return air is dangerous, owing to the quantity of gas given off by the mine, the furnace must be supplied with air fresh from the downcast, and the return air must not come into contact with any fire. When this is the case the return air enters the upcast through a stone drift, termed a *dumb drift*. This drift commences at a given point in the return and rises at such an angle of inclination as to enter the shaft about 50 or 60 feet above the furnace (see fig. 148). The

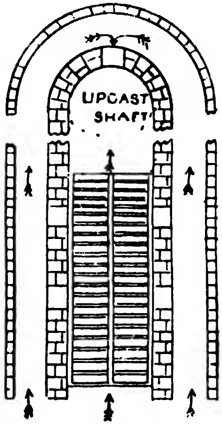


Fig. 146.—Plan of a Furnace

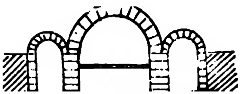


Fig. 147.—Cross-section of a Furnace

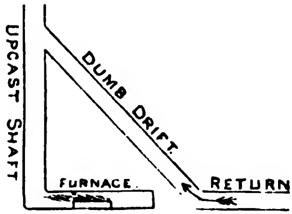


Fig. 148.—Arrangement of Dumb Drift at bottom of Upcast Shaft to prevent Return Air passing over Furnace

Coal Mines Act, 1911, requires that: "Where a fire is used for ventilation in any mine the return air shall be carried off clear of the fire by means of a dumb drift or air-way, unless the mine is one in which inflammable gas is unknown". The Act also prohibits the adoption of furnace ventilation for any new mine started since the passing of the Act.

Furnaces are made from 6 to 12 feet wide, and from

6 to 20 feet in length. When made more than 6 or 8 feet in length they are arranged to be fired at the sides as well as the front. The height above the fire-bars is usually from 4 to 6 feet, and from the bars to the floor about 4 feet.

The quantity of air-current produced by a furnace varies as the square root of the depth of the upcast. For example, a furnace producing a current of air of 50,000 cubic feet per minute, at a depth of 50 fathoms, will produce 100,000 cubic feet per minute at a depth of 200 fathoms. The dimensions of the furnace and the quantity of coals burnt per hour or per day depend upon the depth of the mine and the amount of ventilation required.

Mechanical Ventilators.—These are sometimes applied to exhaust air from the upcast, and sometimes to blow or force air into the downcast. There are two varieties, the varying capacity or displacement machines, and the centrifugal fans.

VARYING CAPACITY MACHINES.—There are two kinds: those which act by pumping the air, as the *Struve* and the *Nixon*; and those which sweep out a definite quantity at each revolution, as the *Fabry*, *Lemielle*, *Cooke*, and *Roots*. They are very rarely applied, because they give a low useful effect, and are very liable to derangements owing to being so complicated in construction.

CENTRIFUGAL FANS.—Practice has shown these to be the best mechanical ventilators, and they are now very largely used. There are many makes, those most generally adopted being the *Guibal*, *Waddle*, and *Schiele*. The *Capell* and *Sirocco* are more recent than these, and give very good results.

Exhaust fans are placed at the top of the upcast, not immediately over the shaft but a few yards away, so that in the event of an explosion they will not be damaged. The Coal Mines Act, 1911, requires that: "Where a mechanical contrivance for ventilation is used at any mine, it shall not be placed beneath the surface". It

is provided, however, that mechanical contrivances may be employed underground when they are auxiliary to the mechanical ventilators placed on the surface.

The upcast shaft is covered at the top to prevent any surface air getting in, and a drift is made to the fan. The action of a centrifugal fan is this: When the fan revolves, the air in it is carried around but does not continue to revolve with the fan, but flies off at a tangent from the circumference. As the fan revolves, the air which enters at the centre moves towards the circumference, where it flies off in accordance with the laws of motion, producing a partial vacuum in the centre of the fan, to which more air rushes, thus keeping up a continuous current. This exhausts the air in the upcast, makes it lighter, upon which the heavier air in the downcast descends and flows towards it.

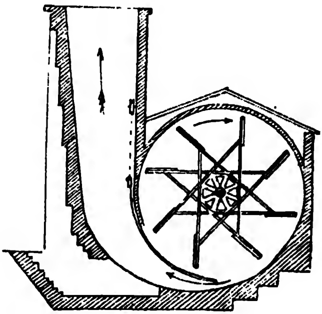


Fig. 149.—Guibal Fan

The *Guibal Fan* is made of all diameters up to 50 feet; the width usually being about one-third of the diameter. It revolves in a casing, and the air, which can enter it at one or both sides, is discharged at one particular place through an adjustable shutter, which can regulate the size of the opening into an expanding chimney (see fig. 149). The blades of the fan are inclined backwards and curved at the tips. It gives very good results, the useful effect varying from 40 to 65 per cent.

The *Waddle Fan* discharges the air all round its circumference into the outside atmosphere, and is, therefore, termed an open running fan. It receives its air at one side only at the centre, and the air passes through channels which have a gradual decreasing section from

the centre to the circumference (see fig. 150). The area of the air-passage at any point between the centre and the circumference, multiplied by the velocity of rotation at the same point, should give a constant quantity. This decreasing section has been adopted to prevent any re-entries of air from the outside atmosphere. It is made of large diameter; several 45 feet diameter are now at work and give good results. The blades are inclined backwards, are cased in at the sides, and form one revolving piece.

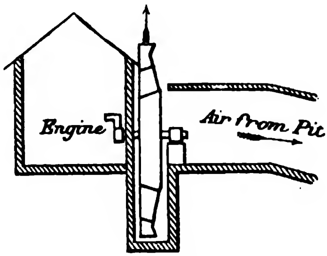


Fig. 150.—Waddle Fan

The *Schiele Fan* is made of small diameter, and revolves at a high speed, from 150 to 300 revolutions per minute. It is not so much used as either of the former. The blades of the fan diminish in width from the centre to the tips, and they are inclined backward, but the tips are *curved* backwards (see fig. 151). It runs in a fixed casing of wrought iron, which is in the form of a spiral, leading into an expanding chimney, so that the air does not enter the outside atmosphere at a high velocity, but has a gradually reduced speed as it ascends the chimney. The fan is usually driven by a belt from an engine, the latter having a large, broad fly-wheel for the belt, and the fan a small one, so that the required velocity is given to the fan without the engine having to run quickly.

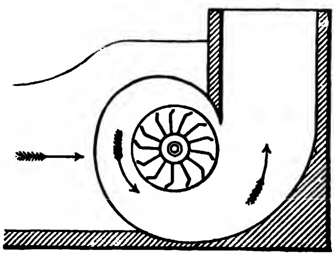


Fig. 151.—Schiele Fan

The *Capell Fan* is made of small diameter, and has single or double inlets to allow the air to pass in on one or both sides from the upcast shaft. It consists of an inner cylinder, in which port-holes are cut, and to this cylinder wings are attached on both sides, those on the outer side being continued to the periphery of the fan, and the whole revolving on one shaft. The air enters the cylinders and escapes through the port-holes into the outer wings, and is carried by them until it is discharged into the évasée chimney. (See fig. 152.)

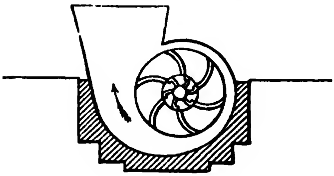


Fig. 152.—Capell Fan

Fans of more recent introduction are the Parsons Turbo - Fan and the Sirocco. Both are of comparatively small diameter, and give exceptionally good results. The former is driven by a steam turbine, and the latter usually by electric motor.

Fans must all be so constructed that it is possible to convert them from exhausting fans to compressive or forcing fans, so that the direction of the air-current can be reversed in the event of an accident rendering such a proceeding necessary, e.g. a fire taking place at the mouth of the downcast shaft.

The accompanying illustration shows the method adopted for reversing the air-current in the Sirocco fan (fig. 153).

Fans of large diameter, such as the Guibal and the Waddle, are well adapted to pass large volumes of air, but they cannot produce high depressions or water-gauges; hence, they are most usefully employed in ventilating mines where the areas of the air-ways are large and the resistance low. On the other hand, small fast-running fans, such as the Schiele and Sirocco, can produce greater depressions but cannot pass such large quantities of air, so that their greatest efficiency

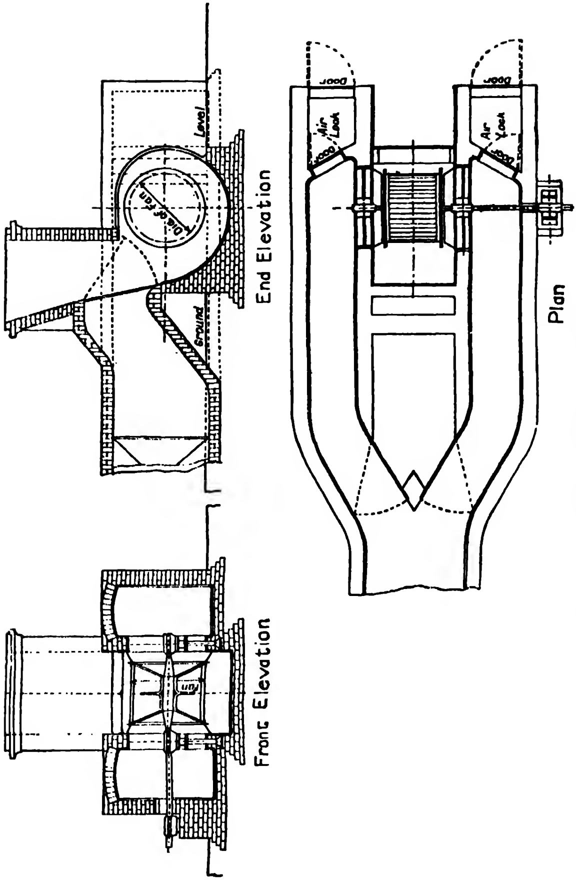


Fig. 153.—Method of Reversing Sirocco Fan

is generally obtained in mines where the air-ways are of small sectional area and high resistance.

The efficiency of a fan is the ratio of the work in the air to the work put into the fan.

EXAMPLE.—A mine fan produces 66,000 cubic feet of air per minute at a water-gauge of 3 inches. Find its efficiency if the brake horse-power of the fan engine when running at the driving speed of the fan is 62·4.

$$\text{Efficiency} = \frac{\text{work given up}}{\text{work put in}} = \frac{66{,}000 \times 3 \times 5{\cdot}2}{62{\cdot}4 \times 33{,}000}$$
$$= {\cdot}5 \text{ or } 50 \text{ per cent.}$$

INSTRUMENTS USED FOR MEASURING VENTILATION, ETC.

The Barometer.

Its Construction.—The ordinary barometer consists of a glass tube nearly 3 feet long, closed at the top end and open at the bottom. It is filled with mercury and inverted, its open end being placed in a basin of mercury. A column of mercury is thus supported by the atmospheric column that presses on the mercury. Placed alongside the glass tube is a scale of inches and parts of inches, fitted with a sliding vernier, by means of which the height of the mercury in the tube may be measured to the $\frac{1}{100}$th part of an inch. The height of the mercury in the glass tube varies from 28 inches to 31 inches, according to the pressure of the atmosphere—the greater the pressure the higher will the mercury ascend in order to balance it. The pressure of the atmosphere varies almost continually, as shown by the rising and falling of the barometer.

To ascertain the pressure of the atmosphere per square inch, multiply the height of the mercury in the barometer by ·4908 lb.—the weight of a cubic inch of mercury. For example, if the barometer reads 30 inches, then $30 \times {\cdot}4908 = 14{\cdot}7$ lb. pressure per square inch. Water is 13·59 times lighter than mercury, therefore it will require a column of water 34 feet in height to balance the atmospheric pressure of 14·7 lb. This is the height to which water can be raised by the common pump, which depends on the same principle as the barometer. Thus, when the piston of the pump is

drawn up, the pressure of the air upon the surface of the water forces water up the pump.

Use of the Barometer.—The barometer is used at mines to show the fluctuations of the atmospheric pressure. This is very important, because a reduction of atmospheric pressure causes the gas in goaves to expand, and it may find its way into the workings and roadways. When, therefore, the barometer falls quickly, greater vigilance should be exercised, especially in fiery mines where there is a large area of goaf. Gas, being very much lighter than mercury, responds even more readily to atmospheric change than the mercurial barometer, and escape of gas in the mine, indicating a change in the atmospheric conditions, may sometimes be observed much earlier than such change is recorded by the barometer.

The barometer may also be used to ascertain the depth of a mine.

The Thermometer.

Its Construction.—An ordinary thermometer consists of a glass tube, closed at the top end and free of air, and having a bulb at the bottom end filled with mercury. A graduated scale of degrees for ascertaining or measuring temperature is placed by its side. Thermometers have not all the same standard of measurement, but are graduated according to three scales, viz. Fahrenheit's, which is generally used in this country, and has the freezing-point marked at 32°, and the boiling-point at 212° above zero; the Centigrade, in which 0° indicates freezing-point, and 100° the boiling-point; and the Réaumur, in which 0° indicates freezing-point, and 80° boiling-point.

Use of the Thermometer.—It is used at mines in conjunction with the barometer to denote atmospheric changes, which have an important bearing on mine ventilation. It is also used to measure the difference of the temperature of the air in the downcast and upcast shafts, where there is natural ventilation or where a

furnace is used. In mines liable to fires from spontaneous combustion it is much used to give warning of any abnormal increase in temperature at any point.

The Hygrometer.

Its Construction.—This instrument consists of a pair of thermometers, one of which has its bulb covered with muslin, kept continually wet by a filament of thread leading to a small vessel of water, and acting by capillary attraction; the other having its bulb dry in the usual way. This is termed a dry and wet bulb hygrometer.

Use of the Hygrometer.—It is used for estimating the dryness of the air. The dryness of the air in mines depends upon the temperature and the freedom from water. As the temperature increases with the depth, deep mines are warmer and dryer than shallow ones. The difference in the temperature of the wet and dry thermometer shows the amount of cold produced by evaporation, and this indicates the dryness of the air. With a moderate temperature of the air, a difference of 2° or less indicates that the air is damp.

Water-gauge.

Its Construction.—This is very simple in its construction and action. It consists of a glass tube of small diameter, bent in the form of the letter U, and is open at both ends. Water is poured in to fill the bend and rise a little up each side, and between the legs is placed a sliding scale of inches and decimals. When greater pressure is applied upon the water on one side than on the other, the water rises up the side of the tube on which the pressure is least, and the difference in level of the two sides may be then measured by the scale (see fig. 154).

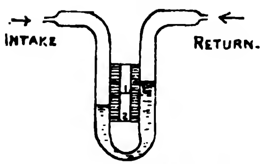


Fig. 154.—Water-gauge, showing difference of Pressure of Intake and Return Air

Since 1 cubic foot of water weighs 62·5 lb., the weight

of a column of water 1 inch high and standing on a base of 1 sq. foot will be $\frac{62 \cdot 5}{12} = 5 \cdot 206$ lb. Hence, when 1 inch of water column is supported by a difference of pressure between the two legs of the water-gauge, that pressure is 5·206 lb. per square foot, and generally water-gauge in inches $\times$ 5·2 = pressure in pounds per square foot.

EXAMPLE.—A fan produces a water-gauge of 4 inches when running at a certain speed on a particular mine. What is the pressure of the ventilating current?

$$WG \times 5 \cdot 2 = p,$$
$$4 \times 5 \cdot 2 = p;$$
$$\therefore p = 20 \cdot 8 \text{ lb. per square foot.}$$

Use of the Water-gauge.—The water-gauge is in use at nearly all mines to ascertain the difference of pressure between the downcast and upcast air-currents, or between the intake and return currents in the mine usually near the shafts. The downcast or intake current, having the greater pressure, presses down the water in the leg exposed to it, and forces up the water in the side of the tube which is exposed to the lesser pressure of the return current. The difference in height of the water is often termed the " water-gauge ", and is expressed by the letters W.G., and from this the ventilating pressure per square foot is calculated.

If the water-gauge varied considerably in its reading between one day and the next, it might be owing to a fall of stone or other obstruction in some of the main air-ways, whereby the area and quantity of air may have been reduced and the friction on the air-current increased. The water-gauge thus indicates to some extent the condition of the air-ways.

The Anemometer.

Its Construction.—This is simply a small fan, the vanes of which are set in motion by the current of air, and the number of whose revolutions is registered upon the face

of a dial fixed on the central part of the instrument (see fig. 155). In some instruments one revolution of the vanes corresponds with one lineal foot traversed by the air.

Use of the Anemometer.—This instrument is used to denote the velocity of the air-currents travelling along the air-ways in mines, by means of which the quantity of air may be calculated in cubic feet per minute. The usual method of using it is to hold the instrument in the air-way for one minute, and read on the dial the register of the velocity or the number of feet travelled by the air in that time. This velocity is then multiplied by the sectional area of the air-way at the place where the velocity is obtained, and the product will be the quantity passing in cubic feet per minute.

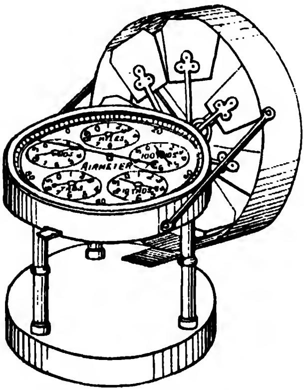


Fig. 155.—Anemometer

Another method of measuring air-currents, formerly much practised, was to explode a small quantity of gunpowder, and to observe the time required by the air to carry the smoke a certain distance along an air-way of equal sectional area between the point of explosion and the point of observation. This gave the velocity, by which the total quantity of air per minute was calculated.

EXAMPLES.—1. The velocity of an air-current measured by an anemometer is 10 feet per second, and the dimensions of the air-way 8 feet × 6 feet; find the quantity of air in cubic feet passing in one minute.

Velocity per minute	=	10 × 60 =	600 feet.
Area of air-way ..	=	8 × 6 =	48 feet.
Quantity per minute	=	600 × 48 =	28,800 cubic feet.

2. The quantity of air passing along a drift 6 feet × 6 feet is 20,000 cubic feet per minute; what is the velocity?

$$\text{Area of air-way} = 6 \times 6 = 36 \text{ feet.}$$

$$\text{Velocity per min.} = \frac{20{,}000}{36} = 555{\cdot}5 \text{ feet.}$$

3. In an air-way 5 feet × 7 feet, if 4500 cubic feet of air pass along in 33 seconds, what quantity will pass in one minute, and at what velocity?

$$\text{Area of air-way} = 5 \times 7 = 35 \text{ feet.}$$

$$\text{Quantity per min.} = \frac{4500 \times 60}{33} = 8181{\cdot}81 \text{ cubic feet.}$$

$$\text{Velocity per min.} = \frac{8181{\cdot}81}{35} = 233{\cdot}7 \text{ feet.}$$

In getting the velocity of an air-current from the explosion of a small quantity of gunpowder, or the burning of a bit of fuse, the following formula may be adopted:

Let V = velocity of the air in feet per minute.
,, D = distance smoke travels in feet in T seconds.
,, T = time in seconds smoke takes to travel distance D.

Since 1 minute = 60 seconds,

$$V = \frac{D \times 60}{T};$$

or if Q = quantity of air passing in c. ft. per minute,
and A = area of mean cross-section of the airway,
then since $Q = V \times A$, we have

$$Q = \frac{D \times 60 \times A}{T}.$$

Example.—An airway is 8 feet wide and 6 feet high. Smoke travels along a distance of 30 feet in 12 seconds; find the quantity of air passing in cubic feet per minute.

$$Q = \frac{30 \times 60 \times 8 \times 6}{12} = 7200 \text{ cubic feet per minute.}$$

CHAPTER XXIV

THE LIGHTING OF MINE WORKINGS

Candles and Open Lamps. Introduction of Safety Lamps. Construction of Safety Lamps. Description of Original and Modern Safety Lamps. Electric Lamps. Firedamp Detectors. Cleaning, Trimming, Locking, Examining, and Testing Safety Lamps.

Candles and Open Lamps.—In mines where no firedamp is given off, open lights, such as candles and lamps, may be used. Near the shafts, and other places where a large amount of light is essential, large stationary oil lamps or illuminating gas lights are used, the gas in the latter case being usually brought from the surface. Persons who work upon the haulage roads usually carry small open lamps, but at the face of the workings, especially in some parts of England, small candles are also used. They give a good light with very little smoke.

Introduction of Safety Lamps.—Where firedamp is given off, the use of candles and open lamps is unsafe. In some mines, where gas is given off in one seam, or part of the workings in one seam, and not in another, safety lamps are used where the firedamp is found, and candles where it does not exist. This mixed system of lighting is not recommended. In other mines open lights are allowed to be used on the main intake airways to a certain point, and beyond that point locked safety lamps must alone be used in the workings and returns. In other mines the quantity of firedamp given off is so great that no open lights are allowed in the mine at all, not even to light up the bottom of the downcast shaft.

When open lights were used in all mines, whether giving off inflammable gas or not, before the invention of safety lamps, explosions were very frequent, and in consequence of the great danger attending the use of

open lights various attempts were made to procure a lamp or other light-giving appliance which would be incapable of igniting firedamp. One of the inventions for this purpose was called the *steel mill*, and was introduced about the year 1733 by Carlisle Spedding. A sharp edge of a flint was applied to a rapidly revolving steel wheel, which produced a shower of sparks, by the light from which the miner had to work. It, however, proved to be very unsafe, the temperature of the sparks being sufficient to ignite firedamp. Nothing better was discovered until about the year 1815, when Davy, Clanny, and Stephenson each produced a lamp which they claimed would not ignite firedamp, yet would give sufficient light for the miner.

What is a Safety Lamp?—A safety lamp may be defined as one that will give light sufficient for the requirements of miners in travelling and working, and will not ignite an explosive mixture in which it may be placed. The two essential merits required in a lamp are therefore—first, safety; and secondly, good light.

A very large number of lamps have been introduced since the invention of the original safety lamps, each claiming superiority over others, but none of them have been found to be perfectly safe and secure under all circumstances when exposed to explosive mixtures. It has been proved that lamps which are considered safe in a slow current become unsafe when the current travels against them, or they against the current, above a certain speed. Mines, nowadays, are much more extensively opened out than in former years, and in consequence the ventilating currents are greater, and move at high velocities. The Davy, Clanny, Stephenson, and many other lamps, which were formerly considered safe, are on this account deemed unsafe in modern mines. Where safety-lamps are used they should be so constructed that they may be safely carried against the air-current ordinarily prevailing in any part of the mine in which they are in use, even though such current should

be inflammable. Safety-lamps must be of a type for the time being approved, as respects the class of mines to which the mine belongs, by the Secretary of State.

The velocities of currents at which the following lamps become unsafe, according to accepted measurements, are:

Davy	6	feet	per	second
Clanny	8	,,		,,
Stephenson	13	,,		,,
Mueseler	21	,,		,,
Bonneted Mueseler	40	,,		,,
Marsaut	40	,,		,,

The last-named type of approved lamps, and some others of more recent invention, being able to safely withstand

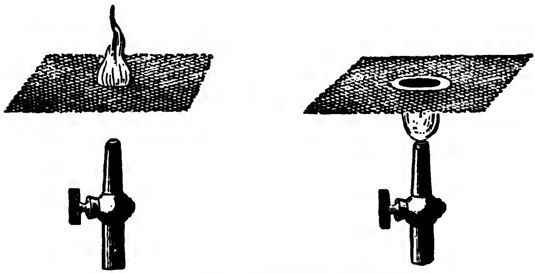


Fig. 156.—Action of Wire Gauze on Flame

such high velocities, which are rarely exceeded in mines, are now largely used.

It may be remarked that nearly all the safety-lamps that have been produced are constructed on the principle of the Davy lamp. Sir Humphry Davy discovered by experiment that the flame of ignited gas would not pass through iron wire gauze having 784 apertures or meshes to the square inch, formed by the crossing of 28 parallel wires per inch of about $\frac{1}{60}$ of an inch diameter (see fig. 156). This is owing to the cooling action of the wire, which reduces the temperature of the flame below that required to ignite the gas. But when the gauze

becomes heated to redness, or when the current in which the flame is placed moves at the rate of 6 feet or more per second, the flame will pass through at a temperature high enough to ignite the gas on the other side. The lamp is accordingly safe only when the gauze is kept cool, in good condition, and when the velocity of the current is under 6 feet per second.

Description of the Original Safety Lamps.

Davy.—This lamp consists of an oil vessel, to which is screwed a brass frame in which is placed an iron wire gauze cylinder about 1½ to 2 inches diameter and 6 inches long. The cylinder is doubled at the top by means of a gauze cap, which affords additional protection from the hot gases which ascend from the flame, and rapidly burn the top of the cylinder. The brass frame consists of a screwed rim at the bottom and three upright poles, which are placed at equal distances apart, and connect the bottom to the top rim. A metal roof or tile is attached to the top rim, and in the middle of it is inserted a handle for holding the lamp. Through the oil vessel, in a small tube, is placed a pricker for the adjustment of the wick. A screwed lock is attached to the bottom rim for locking the oil vessel to the upper portion of the lamp, in order to prevent any person unscrewing it and exposing an open light in the mine (see fig. 157). The Davy lamp gives a very poor light.

Clanny.—This is the oldest lamp having glass in its construction. It consists of an oil vessel screwed to an iron or brass frame. It differs from the Davy in having the flame surrounded by a short thick glass cylinder, above which is a short gauze cylinder (see fig. 158). The object of the glass cylinder is to admit of a good light. The supply of fresh air to feed the flame enters through the gauze at the top of the glass cylinder, passes down on the inside of the glass, and the products of combustion ascend and escape through the upper part of the gauze. This lamp gives a better light than the Davy.

Stephenson. — This lamp is often called "The Geordie". It differs very materially from the Davy and Clanny. The gauze cylinder is made the full length, as in the Davy, but of much larger diameter,

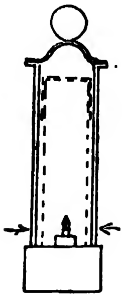


Fig. 157.—Davy Lamp

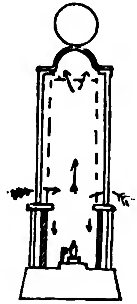


Fig. 158.—Clanny Lamp

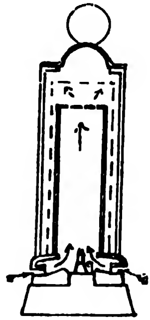


Fig. 159.—Stephenson Lamp

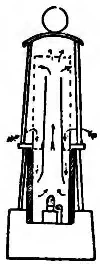


Fig. 160.—Mueseler Lamp

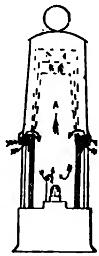


Fig. 161.—Marsaut Lamp.

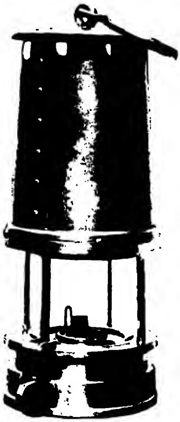
In these lamps the dotted lines are wire gauze; the parts shaded with oblique lines are glass; the double lines at the side are poles; and the bottom part is the oil vessel. The arrows indicate direction of feed air.

and inside of it is placed a glass cylinder reaching nearly to the top of the gauze cylinder. The top of the glass is covered with a perforated copper cap. The feed air enters through very small holes in the bottom rim of the lamp frame, and frequently they become clogged with coal dust. The air ascends through the

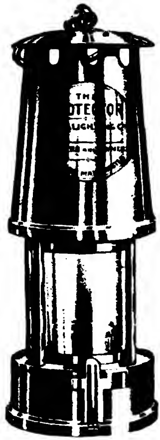
Clanny Lamp with ordinary screw lock

Davy Lamp with ordinary screw lock

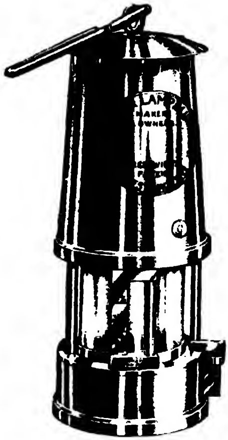
Stephenson's Lamp with screw lock



Marsaut Safety Lamp with lead-rivet lock



Double-glass and Double-lock Lamp. Lit by Low Tension



Gas-testing Lamp. Edmonson-Garton Patent

SAFETY LAMPS
WITH OLD AND MODERN LOCKING ARRANGEMENTS

lamp and passes out at the top (see fig. 159). It causes excellent combustion; but as the light has to pass through both glass and gauze it is very dim on the outside. In combining the glass tube with the gauze the danger of the current forcing the flame through was minimized, but experiments have found it to be unsafe in a velocity of 13 feet per second and upwards. Thousands of Stephenson lamps were in use previous to the passing of the Act in 1887.

Description of Modern Safety Lamps.

Approved Lamp.—No safety lamp shall be used by any person employed in the mine unless it is provided by the owner of the mine, and is of a type for the time being approved by the Secretary of State.

A type to be "approved" must pass a Government test in respect to its design, strength, and general character of construction, and the following requirements must be met, viz.:

In the case of a Flame Safety Lamp

It must be provided with double gauzes of not less than 28 S.W.G. with 28 meshes to the lineal inch (784 to the square inch).

It must be so constructed that it will not be possible to put together the component parts without the gauze.

It must be provided with an efficient locking device. Screw locks not allowed.

It is required to give a minimum candle power of 0·30 during a period of ten hours.

When tested in an explosive mixture in a still atmosphere it must not ignite the mixture exterior to the lamp. When tested in an explosive mixture at a maximum velocity of 1200 feet per minute for two minutes it must not ignite the mixture exterior to the lamp.

Mueseler Type.—This is of the Clanny type with two additions, namely, a horizontal diaphragm of gauze and a conical metal tube or funnel. The diaphragm is placed on the top of the glass cylinder, and supports the

conical or Mueseler tube, and must form a secure and flame-tight joint. The tube is placed in the centre of the lamp a little above the flame, and is about an inch in diameter at the base, and three-eighths of an inch at the top. The air to feed the flame passes through the lower portion of the gauze cylinder, then through the diaphragm down to the flame. The products of combustion pass up the conical chimney, and then through the gauzes. The part of the lamp above the glass is covered with a bonnet or shield of steel with a securely fastened crown to which the carrying ring is attached. Outlet tubes are provided at the top of the bonnet immediately below the crown. The oil vessel is made of iron or steel of sufficient capacity to comply with the requirements and provided with a locking arrangement (fig. 160).

Marsaut Type.—The general design comprises a double gauze above the glass cylinder, each gauze being formed to fit flanges of the outside and inside base rings to make flame-tight joints, and to hold the glass firmly in position in such a manner as to prevent the lamp from being put together without both gauzes, and with top and bottom asbestos washers to ensure flame-tight joints. The oil vessel is made of iron or steel, is of sufficient capacity to comply with the requirements, and is fitted with a suitable locking arrangement (fig. 161).

The gauzes are covered with a bonnet or shield of iron or steel with a securely fastened crown. Outlet holes are provided at the top of the bonnet immediately below the crown.

The feed air passes through apertures in the pillar ring above the glass, through the gauzes, and descends to the flame. The products of combustion ascend in the lamp, and pass through the gauze top and through the outlet holes.

The Marsaut type of lamp is very largely used in gassy mines, probably more than any other type. It is

considered a very safe lamp for general use in mines, and experience confirms this. It has good illuminating power, which is a great advantage in a safety lamp. A very large percentage of underground accidents are due to falls of stone from roof and sides. It is therefore of great importance that a good light should be obtained, so that those in the mine may have the chance to detect faulty places in the roof and sides of the excavations.

Electric Lamps.—The electric light is now largely used for illuminating the heapstead, engine-houses, and other surface works; also in sinking shafts, and at shaft bottoms where the absence of firedamp renders it safe to do so. The fixed electric light and the necessary network of wires are, however, unsafe and inapplicable to haulage roads and the working places, but portable electric safety lamps, complete in themselves, have been invented for use at the working face. They are incandescent lamps, and a storage battery to supply electricity to keep the light burning for a certain number of hours forms part of each lamp. Such lamps are in use at a few collieries, but they are more costly than ordinary safety lamps both in first cost and maintenance.

It should be noted that, as electric lamps are entirely independent of the outside atmosphere, the light being produced in a vacuum, they give no warning of the presence of firedamp or stythe, and on this account many object to their adoption.

Firedamp Detectors.—Another important use to which safety lamps are put, in addition to giving security against explosive mixtures and affording light, is to indicate the presence of firedamp and stythe. The practised observer, by referring to the flame of his lamp, can tell by its behaviour whether the atmosphere is pure or whether it contains firedamp or stythe. If it contains firedamp the flame will be elongated when the lamp is first introduced to the gaseous mixture. If, when this elongation is noticed, the flame of the lamp is

lowered until there is merely a blue rim, with a yellow speck of flame showing on the top of the wick, a blue "cap" of burning gas will appear over the top of the lowered flame. If stythe be present, the light will be dimmed or extinguished. Perhaps one reason why the Davy lamp has been preferred at so many collieries, particularly for the use of officials, is because of its sensitiveness to firedamp.

From 2 per cent upwards of gas in air will show a blue cap above the lowered flame of the safety lamp, which becomes more pronounced as the percentage increases. All firemen, examiners, and deputies must pass a test with percentages of gas from 2 per cent upwards, and no candidate will pass who is unable to see a 2-per-cent cap or make correct observations of gas caps.

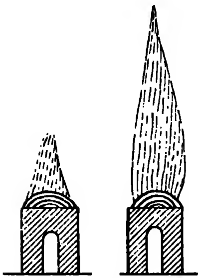


Fig. 162.—Caps above Ordinary Flame indicating Firedamp

In fig. 162 the thin lines indicate the cap of light-blue flame which becomes visible above the flame of the lamp wick when gas is present. In the ordinary method of examination the flame of the safety lamp is reduced by drawing down the wick until the flame is about $\frac{1}{10}$ of an inch or less in height. The lamp is then cautiously raised towards the roof, or into a cavity where gas may be expected, and if present in a small quantity a slight halo will appear above the flame; any increase in the length of the halo or cap, as shown in fig. 162, indicates a larger percentage of gas in the air. Great care is required when raising the lamp; otherwise the gauze may suddenly become filled with flame caused by the ignition of gas inside the lamp. Should this take place the light is at once extinguished.

Although it is extremely easy for the examiner to detect the presence of gas, there are several circum-

stances which make it difficult for him to determine with any degree of accuracy the proportion or percentage of gas present. One examiner may have no difficulty in observing the cap and noting its dimensions, whereas another one may fail to see it until it is more elongated; consequently the first would report the presence of gas, and the other would report no gas. To determine the percentage of gas present from the size and appearance of the cap, correct observations are necessary, and for this the examiner must have training and experience.

The kind and quality of oil used in the lamps influence to some slight extent its sensitiveness as a detector and measurer of the percentage of gas. Probably the best oil to use for this purpose in ordinary safety lamps is colza; other oils may give an aureole above the flame, owing to the volatile spirit they contain, although no gas be present. Such an aureole is known as a fuel cap, and is easily distinguished from a gas cap by the practised observer.

Other conditions, such as the composition of the atmosphere containing the firedamp, and the composition of the firedamp itself, influence the detection of the presence of firedamp. If carbon dioxide is present in the atmosphere, the lamp may not indicate the presence of firedamp which may be there in large quantity. If the firedamp is rich in the higher paraffins it is more readily detected.

Messrs. Cadman and Whalley, in their report on this question to the Royal Commissioners, show that the average mine official cannot detect less than 2 per cent of gas unless he exercise great care. The detection of 1 per cent or less of gas is comparatively easy when an ordinary safety lamp burning a light oil is used. The observer must, however, be trained to the work and be the possessor of good eyesight. Many attempts have been made to add gas-detecting devices to ordinary safety lamps, but none have come into general use.

Cleaning, Trimming, Locking, Examining, and Testing Safety Lamps.

In accordance with the provision of the Coal Mines Act, 1911, Rule 33, " No safety lamp shall after the 1st day of January, 1913, be used by any person employed in the mine, unless it is provided by the owner of the mine, and is of a type for the time being approved, as respects the class of mines to which the mine belongs, by the Secretary of State ".

The owners of the mine are therefore responsible for providing safety lamps, and also cleaning, trimming, and maintaining them.

Cleaning and Trimming.—As soon as lamps are deposited in the lamp-room after being used in the mine, they are at once examined as to their condition in case they have been subject to rough usage and damaged. They are then unlocked, unscrewed, and each part separated for cleaning. Cleaning the parts may be done either (1) by hand labour, or (2) by machine.

(1) *By hand labour.*—If there are not a large number of lamps to deal with, the unscrewing, &c., is done by men or boys. The latter then carefully brush both the outside and inside of the lamp-gauzes, and at the same time examine them to see that none of the wires have been damaged or the meshes enlarged. The glasses are also cleaned outside and inside and any damaged ones rejected.

(2) *By machine.*—Where there are a large number of lamps to deal with at various times both day and night a lamp-cleaning machine is generally used. This consists of a number of brushes of suitable size and shape, which are kept revolving by belting usually driven by a small engine. Small circular brushes are provided for quickly cleaning the inside of the gauze. Fittings are attached to the machine to facilitate the work of separating the various parts of the lamps.

Trimming the oil vessels of the lamps is an important operation. This consists in providing them with suit-

able wicks, and also re-filling with oil. As the light with which the miner has to work depends upon the satisfactory oiling and trimming, it is important that this work should be done carefully.

The oil used should be of such a kind as not to readily give off inflammable vapours when it becomes heated in the mine. Light oils, such as petroleums, are unsuitable for use in safety lamps unless the latter are specially designed. The light yielded is superior to that of heavy oils, and is now largely used.

Locking.—It is essential for the safety of the mine that the safety lamps in use be so constructed and locked that the light cannot easily be reached and exposed to the mine atmosphere by any person wilfully attempting to unlock a lamp. Also the locks should be so arranged that any attempt to unlock a lamp, by any workman without authority, shall not escape detection.

Methods of locking lamps:

1. By means of a small screw bolt which works through a head fitted on the bottom rim of the lamp, and is turned until the bottom rim and the oil vessel are secured together by means of the bolt pressing against the latter. This method is now prohibited.

Owing to its simplicity of construction, and the readiness with which lamps could be locked and unlocked, this arrangement has in the past been largely adopted. The ease by which lamps could be unlocked constituted, however, its weakness, as a workman could contrive a key to open his lamp, and also lock it again, without leaving any trace whereby he could be detected.

2. By means of a riveted lead plug which passes through an eye and fixes the lamp body and the oil vessel together.

The lead plug is put through the eye and then firmly riveted, and each end of the plug stamped with some distinguishing mark.

This method of locking offers no greater difficulty than the screw bolt to any workman wishing to sur-

reptitiously open his lamp. Its advantage lies in the fact that he could not replace and restamp the lead plug, and therefore the detection of his breach of rules would be certain.

3. By means of a magnetic spring lock. This is self-locking and requires a powerful magnet to unlock, and is no doubt the safest arrangement. In the Wolff lamp a ratchet is pressed by means of a spring against teeth on the oil vessel, and this prevents its being unlocked until a powerful magnet is brought to bear upon it which withdraws the ratchet and allows it being opened.

4. Another arrangement is termed the "protector" lock. A contrivance is fitted inside the lamp which causes the light to be extinguished if the oil vessel is unscrewed. Therefore any workman attempting to unscrew his lamp for the purpose of getting an open light would defeat his object.

Examining and Testing.—Great care is necessary in putting the various portions of the lamp together. The safety of a lamp depends upon the gauzes, the glass cylinder, and the asbestos rings between being properly placed and in good order; also the oil vessel tightly screwed up and the lamp securely locked. After being lighted and locked the lamp should be subjected to strict examination in the lamp-room before it is handed out, and again at the lamp-station before it is allowed to be carried into the workings.

The general way in which this examination is carried out is to scrutinize the lock, then test the lamp bottom to ascertain if it is tightly screwed, and also to blow sharply upon the lamp at the top and bottom of the glass cylinder. If the lamp is satisfactory, the light will not be deflected by this blowing.

At some collieries all lamps when they are lighted and fitted together to be handed out to workmen are put into a gas tester which contains inflammable gas. If the lamp passes this test without showing any defects, it is considered in good order for use underground.

Usually all lamps are numbered, and either each workman is allocated a particular lamp, or the names of each and the lamp numbers are recorded in a book when given out each day. This is necessary, so that a proper record may be kept of the name of each workman and of the lamp he is responsible for during his shift underground.

Electrically-ignited Safety Lamps.—To facilitate the lighting of safety lamps, many are now fitted with a contrivance whereby they can be lighted without unlocking by the switching on of an electric current. When a large number of lamps are required to be handed out at a particular time, it is an advantage to be able to light electrically. It is also an advantage underground if properly safeguarded, and may avoid a good deal of loss of time which frequently occurs owing to lamps being accidentally extinguished at the face of workings. The lamps are taken to the lighting-station, which ought to be under the control of a responsible person, and are lighted without being unlocked. This arrangement is not altogether satisfactory, as, in the opinion of many, no lamp ought to be re-lit until it has been taken to pieces and each part, especially the gauzes, carefully examined.

The ignition is made by the igniter, which, on being brought into contact with an insulated copper ring fixed at the bottom of the glass, induces sparking at the copper wick-tube.

The apparatus should be enclosed in a flame-tight casing, and provided with a suitable switch or safety-plug so that no sparking can occur except between the terminals provided for the purpose inside the safety lamp.

Electric Safety Lamps.—For use underground these must pass the Home Office test, and must comply with the following conditions:

No liquid must escape from the battery when the lamp is turned upside down.

The switch and other electrical contacts must be contained in flame-tight enclosures.

The lamp must be provided with an efficient locking device to prevent unauthorized persons from tampering with the electrical contacts.

The lamp must give a light of not less than 1 candle-power all round in a horizontal plane throughout a period of not less than nine hours.

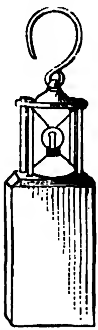


Fig. 163.—The Gray-Sussmann Electric Lamp, as designed for use in Collieries and Mines

The lamp is tested by the light being switched on and off while the lamp is in an explosive mixture.

The light being in a vacuous bulb is without any means of communication with the outside atmosphere. The latter, however, appears to be the chief reason why electric safety lamps do not come into more general use. Being entirely independent of the surrounding atmosphere, which the miner has to breathe and work in, they give no indication of the nature of the atmosphere, whether it is safe to breathe, or whether it contains inflammable or deleterious gases. This is regarded as a grave defect compared with the ordinary safety lamps, which do indicate the nature of the surrounding atmosphere, and by their behaviour warn the miner when either firedamp or stythe is present in the air in a dangerous quantity.

The Gray-Sussmann Electric Safety Lamp is shown in fig. 163. It is approved by the Home Office in accordance with the requirements of Section 33, Coal Mines Act, 1911. It is provided with a sheet-steel case with securely soldered joints, and a hinged stamped-steel cover forming a flame-tight connection with the case and secured by an efficient locking device; an insulated domed base-plate secured to the cover, and carrying the

bulb, switch, and two spring contacts making connection with the terminal lugs of the electrical accumulator. The lamp weighs $5\frac{1}{2}$ lb., and conforms with the Home Office requirements that an electric lamp shall give 1 candle-power at the end of nine hours' use. Another electric safety lamp, which is being largely used, is the "Ceag". It weighs 5 lb. 3 oz., and is rated at 1·5 candle-power for fourteen hours.

No satisfactory appliance has yet been attached to electric lamps for the purpose of estimating and detecting the presence of firedamp, but endeavours are being made to invent an arrangement which will be part of the lamp and indicate certain percentages of gas in air.

CHAPTER XXV

UNDERGROUND CONVEYANCE OF COAL

Construction and Dimensions of Tubs. Underground Railways and their Maintenance. Haulage by Men and Horses. Self-acting, and Balance Inclined Planes.

Construction and Dimensions of Tubs or Trams.—A tub or tram is a rectangular box made of wood, iron, or steel fixed to a wood or steel frame and carried upon two axles and four wheels, and used for conveying coal from the working face to the surface. The wood frame is generally of oak, and a centre or draw-bar of iron or steel of the full length of the frame is fixed underneath and in the centre of the latter. It is usually provided with a hook at one end, and a chain, called a coupling-chain, at the other, for the purpose of coupling two or more tubs. When the body of the tub is made of wood, it is strengthened with straps of iron; when made of sheet iron or steel, the sheets are riveted together, and are also strengthened with straps of iron, and securely bolted to the oak frame. (See fig. 164.)

Corrugated sheets are sometimes used to form the sides above the frame in order to get the necessary stiffness.

The dimensions of a tub are determined chiefly by the height or thickness of the coal-seam. In a high seam the tub must not be made of greater size than can be handled by men, and in a thin seam the tub must be made to suit the circumstances. The capacity of tubs varies from 6 cwt. to 20 cwt. of coal.

Wheels and Axles of Tubs.—The wheels and axles of

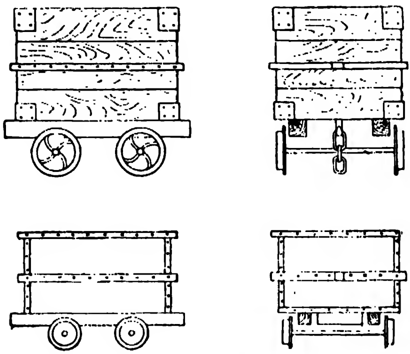


Fig. 164.—Side and End Views of Wood and Iron Coal Tubs

tubs are made of iron or steel, usually the latter. There are two ways in which they are connected relatively to each other.

1. The wheels securely fixed to the ends of the axle, so that the wheels and the axle revolve together.

This method of connection is invariably adopted on surface railways, the conditions being suitable. The roads are made even, regular, and as straight as circumstances will admit. Curves are avoided as much as possible, and where they are absolutely necessary the radius is made as large as possible. Wheels fixed to an axle move together equally in the same direction and with the same velocity. Their tendency is to run in a

continuous straight line; they are therefore unsuitable for use on railways having sharp curves. The main haulage roads in mines nowadays approximate to the perfection of surface railways. They are made as straight, even, and solid as possible, and fixed wheels are largely adopted. Of course the roads in the inner workings near the face are usually rough and irregular, with occasional short sharp curves. These conditions are unfavourable to the adoption of fixed wheels. In passing round the curves there is a great amount of friction produced by the wheels sliding upon the rails, and the flange of the outer wheel grinding against the rail. There is also a tendency of the wheels to jump off the rails on the outer side of the curve; to resist this tendency the outside rail should be raised.

2. The axles fixed to the tub, and the wheels revolve loose at the ends of the axles.

In this method the wheels are completely independent of each other, and may revolve in different directions and with different velocities. This system is invariably applied to road vehicles which run on uneven surfaces in a sinuous path and require to make sharp curves very frequently. The chief advantage of loose wheels is their adaptability for passing round curves, and in some mines, where the roads are unavoidably irregular, uneven, and winding, loose wheels are adopted. There are more repairs to loose wheels, which are kept on the axles with cotters and washers, than to fixed wheels. They rarely run true, the gauge being seldom correct, whereas with fixed wheels the gauge can be accurately secured. Loose wheels appear to be more used in American than British mines.

In some instances combinations of the two methods have been tried, with the view of obtaining the advantages of each. In one of these the arrangement consists of having one of the wheels fixed to the axle and the other loose, so that slipping and skidding may be avoided at curves.

Resistances to Traction.—When a body is moved along a horizontal surface, the resistance to its motion to be overcome is due to friction. The resistance to the motion of tubs is due to the friction of the wheel upon the rails, and to the axle upon its bearings. Theoretically there should be little friction caused by the rolling of the wheel upon the surface of the rail, but in practice the roads are uneven and liable to get covered with dust and dirt, and in consequence friction is considerable. The friction of the axle upon its bearings varies directly as the weight of the tub and the diameter of the axle, and inversely as the diameter of the wheel. If the diameter of the axle be doubled, and the diameter of the wheels remain the same, the friction will be doubled; but if the diameter of the wheels be doubled, and that of the axle remain the same, the friction will be one-half. Thus, if the resistance due to friction = 50 lb., by doubling the diameter of the wheels the resistance will be reduced to 25 lb. Therefore, so far as the limits of strength and the height of the seam will allow, the smaller the axles are made, and the larger the wheels, the less will be the amount of friction.

Experiments have shown that, to a vehicle with ordinary wheels and axles drawn along a good macadamized road, the resistance of friction is about $\frac{1}{32}$ of the weight of the vehicle, or about 70 lb. per ton—that is, a horse, in order to draw a vehicle 1 ton weight, must exert a force of 70 lb. This is called tractive force. To railway wagons on an ordinary surface railway the resistance of friction is about $\frac{1}{224}$, or 10 lb. per ton. To tubs on underground haulage roads the value of the friction varies a great deal, owing to the different conditions that prevail in mines. The average may be taken at about $\frac{1}{70}$, or 32 lb. per ton.

The fractions given above are called coefficients of friction. They are constant numbers found by experiment, and represent the proportion between the pressure and the friction between two particular surfaces

In passing round curves a much greater resistance is met with than on straight roads, especially, as already mentioned, if the wheels are fixed to the axles.

Resistances to traction due to the varying inclination of underground roads have an important bearing on haulage. The roads may be level, dipping, or rising. We have already stated the resistances of friction on level roads. Where the roads are to the dip the inclination is unfavourable to haulage; the empty tubs going inbye will be assisted by gravity, but the full ones coming outbye will be retarded, and the power required to overcome the resistance due to the inclination will be greater in proportion to the angle of inclination.

Where the roads are to the rise the inclination is favourable to haulage, provided the angle is not a steep one. In laying out the main roads advantage should be taken of the dip in the direction of the shaft, so that self-acting inclines may be adopted to reduce the cost of haulage to the lowest point. If the rise inbye is about 1 in 30, or steeper, the tubs will run by gravitation, and a self-acting incline may be adopted. Any inclination less than this is, however, insufficient for such a method. Where self-acting inclines are impossible, the conditions may still be such as will allow of the roads having such an angle of inclination as will make the force of traction or the resistances to the full and empty train of tubs about equal.

Underground Railways and their Maintenance.—The roads or railways on which the tubs travel from the face to the shaft may be divided into three kinds:

1. The temporary or tramroads near the face.
2. The rolley-ways or horse-roads.
3. The self-acting inclines and engine-planes.

1. In the workings the roads are generally circuitous and temporary. The rails used on them are usually of a light description, and are laid down during working

hours as the face advances. Sometimes the rail consists of a flat iron bar about $1\frac{1}{2}$ inches × $\frac{3}{4}$ inch section wedged into notches cut in the sleepers. These notches are cut at such distance apart as will correspond with the width between the wheels of the tubs. Bridge rails, 3 to 6 feet in length, and weighing from 14 to 22 lb. per yard, are more generally adopted. They are nailed to the sleepers upon which they rest. The sleepers are generally of larch, about 3 or 4 feet long × 5 inches × $2\frac{1}{2}$ inches, and placed transversely at intervals of 3 or 4 feet. This kind of railway may be very quickly laid down or pulled up (see fig. 165).

2. Horse roads are sometimes termed rolley-ways and wagon-ways. The rails used upon them are heavier and longer, and more care in laying them is required than in the temporary roads, because trains of from 6 to 12 tubs may be hauled over them. Flat-bottom rails weighing about 24 lb. per yard in 12-foot lengths, and spiked to longitudinal sleepers, form a good horse road. Transverse sleepers require frequent renewal, as they are soon worn out by the horses.

Fig. 165—Bridge Rails

3. Roads upon which the tubs are moved by machinery are termed engine-planes. On these roads it is essential that the railway be laid solid and accurate and kept in a state of efficiency, particularly where the tubs are large and the speed of haulage is high. Curves ought to be avoided, but where they must be made they should have a large radius.

The rails used weigh from 24 lb. to 40 lb. per yard in from 12 to 18 foot lengths. Double-headed rails, with cast-iron chairs upon transverse sleepers, and fastened with wooden pins and keys, are used in some mines. Iron or steel flat-bottom rails, spiked to transverse or longitudinal sleepers, and having the joints of the rails fished, are largely adopted (see figs. 166, 167). The size of the sleepers should be in proportion to the weight

of the rails used. Steel sleepers have been introduced within the last few years, and are used upon engine-planes.

Conveyance of Coal from the Face of the Workings to the Shaft by Men, Horses, Inclines.

By Men.—At a convenient point a short distance from the face of the workings in each district of the mine, sidings, flats, or stations, as they are variously termed, are formed by having a double line of rails for a distance of 30 or more yards. One line is for a train of empty tubs and the other for a train of full tubs to stand on. The empty tubs are taken singly to the various working places by young men, termed *putters* or *trammers*, and when

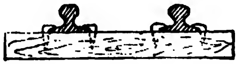


Fig. 166.—Heavy Flat-bottom Rails for Engine-planes

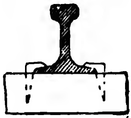


Fig. 167.—Flat-bottom Rails fastened to a Longitudinal Sleeper

filled they are pushed back to the siding. Small ponies are applied to this work when the height of the seam will admit them. They may be yoked into wood shafts or *limmers*, which they carry about, and are readily attached to or detached from the tubs, or they may draw the tubs by means of a tail chain attached to a cross-tree. The latter is the safer method, as accidents occur by the drivers riding on the limmers. This they cannot do when a tail chain is used.

By Horses.—From these sidings the tubs may be taken in trains by horses direct to the shaft bottom if the distance is short, or to a main engine-plane station, from which they are conveyed along the engine-plane by machinery. In some cases the tubs are brought direct from the putting siding or flat to the shaft by mechanical haulage without the intervention of horse haulage.

The number of tubs that a horse can haul depends chiefly upon the gradient. A dip "outbye" or outwards,

in the direction in which the full tubs are going, is most favourable, though, if it be too much, it will be unfavourable to the haulage of the empty tubs. An outward dip of about $\frac{1}{4}$ inch per yard is the most favourable gradient for horse haulage, the resistance to the full and empty trains being equal. The greatest useful effect is obtained from horses when travelling at the rate of about three miles per hour, and pulling steadily.

Let T = tractive force in pounds.
F = friction of load set.
f = friction of empty set.
G = gravity acting on load set.
g = gravity acting on empty set.
W = weight of load set in pounds.
w = weight of empty set in pounds.
I = inclination expressed as 1 vertical to so many along the incline or otherwise as the sine of the angle of inclination.

Then G = W I. Gravity is positive or negative according to whether load is moved uphill or downhill. In order to obtain the best results the inclination should be such as gives equal tractive efforts both ways, i.e.

$$
\begin{aligned}
\text{tractive force in empty} &= \text{tractive force out load,} \\
f + g &= F - G, \\
\text{by substitution } f + wI &= F - WI, \\
\text{hence } I(w + W) &= F - f, \\
\text{or } I &= \frac{F - f}{W + w}.
\end{aligned}
$$

EXAMPLE.—Find the best inclination for a horse road when the tub weighs 4 cwt. and contains 10 cwt. of coal. Take friction as 32 lb. per ton.

$$
\begin{aligned}
\text{Here } F &= \frac{14 \times 32}{20} = 22{\cdot}4, \\
f &= \frac{4 \times 32}{20} = 6{\cdot}4, \\
W &= 14 \times 112 = 1568, \\
w &= 4 \times 112 = 448, \\
\text{and } I &= \frac{F - f}{W + w} = \frac{22{\cdot}4 - 6{\cdot}4}{1568 + 448} = \frac{16}{2016} = \frac{1}{126}.
\end{aligned}
$$

∴ best inclination is 1 in 126 in favour of the load tubs.

Inclines.—When the road is straight, and the dip in the direction of the shaft is 1 in 30 or upwards, the descent of the full tubs is sufficient to haul up at the same time the empty ones. Such an arrangement is called a *self-acting incline*, and is the easiest and most economical method of transit underground. A sheave or a drum with brake attached is fixed at the top of the incline (see fig. 168). Only one rope is generally used, which passes around the sheave, and is attached at one end to the full tubs at the top and to the empties at the bottom. The rails are sometimes laid double from the top to the bottom of the incline, and the full tubs as they run down on one line haul up the empties on the other. In many cases the top half of the incline is laid with three rails, and at "meetings" where the full and empty trains pass each other double lines are laid, then below meetings to the bottom only a single line is used (see fig. 169). This arrangement is very useful where the roads are narrow. The number of tubs forming a train

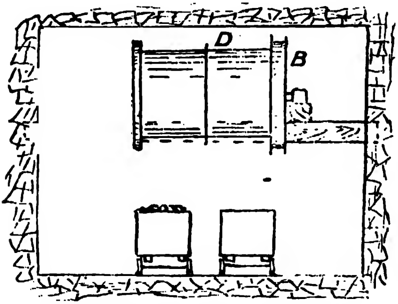


Fig. 168.—Arrangement of Drum for Two Ropes for Incline

D, Drum elevated above tubs. B, Brake for regulating speed of tubs.

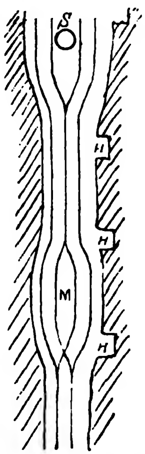


Fig. 169.—Self-acting Incline.

Arrangement of Rails: S, Sheave. M, Meetings. H, Refuge holes.

depends upon the inclination and length of the incline. In some instances, instead of the incline running a train intermittently, an endless rope is used, running continuously, with the tubs attached singly at intervals.

A *jig-brow* is a simple form of incline used in steep seams near the face for running down one full tub at a time to a siding or main road. The wheel round which the rope travels is usually held by an iron frame to a stout prop firmly fixed between roof and floor, and which can be readily removed and advanced. The frame carrying the wheel consists of two plates terminating in a strong bolt which passes through the prop, and is fitted with a strong hand-brake (see fig. 170). Two sets of rails are provided, and sometimes the full tub in descending on one set pulls up the empty tub on the other set; in some cases, however, a balanced bogie on a narrow gauge is used.

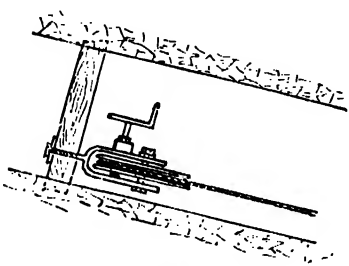


Fig. 170.—Jig-brow Wheel

On the Continent, where very steep seams are being worked, a balance-incline is sometimes adopted. Two carriages are used; one has a horizontal platform, upon which the tub, full or empty, is carried, the other is weighted, and acts as a counterbalance. The full tub upon the carriage descends and hauls up the counterbalance, then the counterbalance descends and hauls up the empty tub. The counterbalance runs on a narrow gauge within the other, and is so low as to pass underneath.

The available power in a self-acting incline is gravity acting on the load tubs, and this gravitational pull must be greater than the sum of all the other resistances involved, including the effect of gravity upon the empty

tubs that have to be raised from the lower to the higher point. Using the same symbols as in the last case, this may be expressed as:

$$G > F + f + g + \text{rope friction} + \text{rope gravity},$$
$$\text{i.e. } WI > F + f + wI + \text{rope friction} + I;$$
$$\text{and } WI - wI - \text{rope } I > F + f + \text{rope friction}.$$
$$I(W - w - \text{rope}) > F + f + \text{rope friction}.$$
$$\therefore I > \frac{F + f + \text{rope friction}}{W - w - \text{rope weight}}.$$

EXAMPLE.—Find the flattest inclination at which an incline will be self-acting when the set consists of 12 tubs each weighing 4 cwt. empty, and 14 cwt. gross when loaded. The rope used weighs 2·5 lb. per fathom, the incline is 300 fathoms long, and friction 32 lb. per ton.

The friction of a rope in cases of this kind is usually taken as $\frac{1}{20}$ of its weight.

$$W = 12 \times 14 = 168 \text{ cwt.} = 18{,}816 \text{ lb.} = 8{\cdot}4 \text{ tons.}$$
$$w = 12 \times 4 = 48 \text{ cwt.} = 5376 \text{ lb.} = 2{\cdot}4 \text{ tons.}$$
$$F = 8{\cdot}4 \times 32 = 268{\cdot}8, \text{ say } 269 \text{ lb.}$$
$$f = 2{\cdot}4 \times 32 = 76{\cdot}8, \text{ say } 77 \text{ lb.}$$
$$\text{Rope} = 300 \times 2{\cdot}5 = 750 \text{ lb.}$$
$$\text{Rope friction} = \frac{750}{20} = 37{\cdot}5 \text{ lb.}$$

$$I > \frac{F + f + \text{rope friction}}{W - w - \text{rope weight}}$$
$$> \frac{269 + 77 + 37{\cdot}5}{18816 - 5376 - 750}$$
$$> \frac{383{\cdot}5}{12690}$$
$$> \frac{1}{33};$$

that is, the inclination must be steeper than 1 in 33. In practice 1 in 27 or so would be required in order to have a margin of force available for acceleration.

CHAPTER XXVI

UNDERGROUND CONVEYANCE OF COAL BY MACHINERY

Systems of Haulage: Main Rope; Main and Tail Rope; Endless Rope; and Endless Chain.

Haulage by Stationary Engines. — Very few mines are so fortunately situated that all the coals gravitate to the shaft and require no haulage by machinery. In most mines the roads are more or less undulating, and when they extend beyond a certain distance horse haulage must be replaced by a cheaper and more expeditious system of engine haulage. The engines used for this purpose are mostly stationary, and are built either on the surface or underground. When on the surface, the ropes are carried down by the shaft side in wooden boxes. Where chains are used in haulage the engine should be placed underground, particularly in deep mines. Steam to an underground engine may be conveyed from the surface, or boilers may be built underground near to the upcast shaft. The latter plan, however, is very objectionable. Steam is rarely taken into the mine now, as electric power transmission is much more convenient and less wasteful of both work and capital.

Systems of Haulage.

Main Rope System or Direct Haulage.—This system may be applied when the engine-plane has a uniform dip of 1½ inches per yard or upwards from the shaft into the station. A rope is attached to the empty tubs at the shaft, and as they run in by the force of gravity they drag the rope with them. At the station the rope is taken off and attached to the full train, and on the signal being given the engine hauls the latter up the hill to the shaft. Only one rope and one drum on the engine is

required; the latter is " thrown out of gear ", or made to run loose on the engine shaft, when the empty train is running in.

EXAMPLE.—Find the power required to draw a train of 8 tubs up an incline of 1 in 5 when the tubs are 14 cwt. gross weight, the rope weighing 4 lb. per fathom, friction 32 lb. per ton, and speed of train 10 miles an hour; the incline is 1000 yards in length.

Here T = F + G + rope friction + rope g

$$= \left(\frac{8 \times 14}{20}\right)32 + \frac{8 \times 14 \times 112}{5} + \frac{2000}{20} + \frac{2000}{5} = 3188 \text{ lb.}$$

Ten miles per hour $= \dfrac{5280 \times 10}{60} =$ 880 ft. per minute.

Work done = 3188 lb. × 880 ft. = 2,805,440 foot-pounds per minute.

Horse-power $= \dfrac{2{,}805{,}440}{33{,}000} = 85.$

Main and Tail Rope System.—In this system two ropes are required: one, called the *main rope*, hauls the full trains out to the shaft; the other, called the *tail rope*, hauls the empty trains in. Two drums are required, one

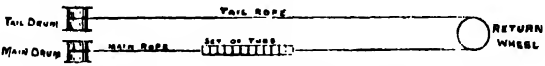


Fig. 171.—Illustrating Main and Tail Rope System of Haulage

for the main and the other for the tail rope. Each drum is on a separate shaft and is connected by spur gearing to the engine shaft, and both are so arranged that when one is put into gear or connected to the engine, the other is thrown out of gear. Each drum is provided with a brake (see fig. 171).

The tail rope is conducted from the drum along the side of the engine-plane to the extreme end, where it passes around a sheave and returns along the centre of the engine-plane to the shaft, where it is connected to the front end of the empty train. The main rope is attached to the other end of the train. When the engine

is started to haul in the empty train, the main rope is drawn off the drum which is out of gear, and the tail rope is wound on the other drum, which is in gear. When the siding or station is reached, the ropes are detached from the empty train and attached to the full train. The main drum is put into gear and the tail drum thrown out of gear; then the main rope hauls out the full train, which drags the tail rope with it. Friction rollers, to keep the ropes from dragging on the floor, are laid in the middle of the engine-plane for both ropes,

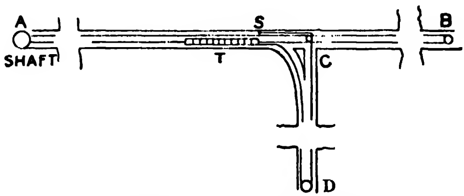


Fig. 172.—Main and Tail Rope Haulage Junction

and at the side for the tail rope, at intervals of about 10 yards.

Where haulage on this system has to be worked with one or more branch roads, arrangements have to be made at the junctions or " wayends " for connecting up the branch ropes to the train of tubs. There are two or three methods for working branch roads; the one shown in fig. 172 is very often adopted. There is a main road AB, and a branch road CD, at right angles to the main road. The empty train T has been hauled " inbye " to the junction C by the main-road tail rope passing round the return wheel at B. This rope is detached from the tubs, and also uncoupled at the " changing socket " S. The tail rope along the branch road passing round the return wheel at D is coupled on at S, and attached to the train of tubs T. The hauling-engine is started, and the empty tubs are drawn into the " landing " or station at D. The ropes are detached from the empty tubs, and

attached to the full ones, and these are hauled out to the shaft. Then the empty tubs, forming another train, are hauled again to the junction C. If the empties are required at the landing at B, the ropes are again changed.

To facilitate the changing of the ropes a hand winch is fixed at the junction to pull the branch rope into position to allow the connection to be readily made, but in any case it is necessary that the engine should run the train to an arranged position at the junction.

The advantages of the main and tail rope system of haulage are:

1. A single line of rails only is required, consequently it can be applied on comparatively narrow roads.

2. Well adapted for long distances, and where there are branch roads.

3. Can be used where bends and undulations exist.

4. Workmen's trains or "sets" for riding "inbye" or "outbye" may be arranged when the "faces" of workings are long distances from the shafts.

The disadvantages of the system are:

1. Quick speed at which the tubs forming the trains have to travel in and out, viz. 6 to 12 miles per hour.

2. Heavy wear and tear of rolling stock in consequence of high speed.

3. Intermittent supply of full tubs at the winding-shaft.

4. Large engines required, as there is no counter-balancing of the tubs on the undulating gradients.

EXAMPLE.—A main and tail rope has to haul 400 tons per day of 8 hours from a distance of 1600 yards. The average speed is 8 miles an hour and the heaviest gradient 1 in 10. The tubs weigh 4 cwt. empty and hold 10 cwt. of coal. Find the number of tubs per journey and the horse-power expended, assuming friction as 32 lb. per ton of load and weight of rope 3·5 lb. per fathom.

$$\text{Average speed} = \frac{8 \times 5280}{60} = 704 \text{ feet per minute.}$$

$$\text{Time running} = \frac{1600 \times 3 \times 2}{704} = 13{\cdot}6 \text{ minutes.}$$

Allow 3·2 minutes for changing at each end, then total time per journey $= 13{\cdot}6 + 3{\cdot}2 + 3{\cdot}2 = 20$ minutes, and $\frac{60}{20} = 3$ journeys per hour; but $\frac{400}{8} = 50$ tons per hour to be drawn, hence $\frac{50}{3} = 16\frac{2}{3}$ tons per journey. Since each tub holds 10 cwt. this means 34 tubs on each journey, and weight $= \frac{34 \times 14}{20}$ $=$ 23 tons 16 cwt., or with couplings say 24 tons.

Rope is 1600 fathoms long (double along 1600 yards of roadway).

∴ Rope weight $= 1600 \times 3{\cdot}5 = 5600$ lb., and rope friction $= \frac{5600}{20} = 280$ lb.

$T = F + G +$ rope f. Rope gravity does not come in as the rope balances.

$$T = 24 \times 32 + \frac{24 \times 2240}{10} + 280 = 6424 \text{ lb.}$$

Speed is 704 feet per minute; ∴ work done $= 6424 \times 704$ foot-pounds per minute,

$$\text{and horse-power} = \frac{6424 \times 704}{33{,}000} = 137.$$

Endless Rope System.—This system of haulage has come very much into use. Two sets of rails are usually required—one for the empties travelling "inbye", and the other for the full ones coming "outbye". The

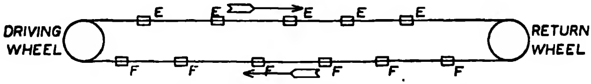


Fig. 173.—Illustrating Endless Rope System of Haulage

E E, Empty tubs going from the shaft into the workings. F F, Full tubs returning to the shaft.

endless rope is moved by a sheave around which it travels continuously in one direction, actuated by suitable gearing from the engine. It travels along the engine-plane, passes around a sheave at the extreme end of the plane and then returns (see fig. 173).

There are three ways in which the endless rope is applied:

1. The rope travels under the tubs.
2. The rope travels over the tubs.
3. The rope travels by the side of the tubs.

1. The tubs are attached singly, or in trains of three or more tubs, to the rope at intervals. The method of attachment is by short chains or by haulage clips. These are fastened to the tub coupling, and are then made to grip the rope (see fig. 174). The simplest arrangement of clip consists of a pair of iron jaws working on an axis and formed to grip the rope. A loose collar, free to slide up and down, is placed around the jaws to keep them together and cause them to grip the rope. They are easily attached, and may be disengaged by a self-acting arrangement.

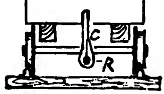


Fig. 174.—Rope under Tubs

R, Rope. C, Clip.

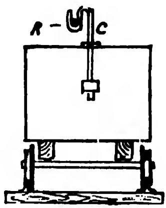


Fig. 175.—Rope over Tubs

R, Rope. C, Clip.

2. This method is not applicable where the tubs are loaded above the level of the top. The tubs may be attached singly, or in trains at intervals, by means of short attachment chains or clips (see fig. 175).

3. This arrangement is not a good one, and is not much used. Where there are curves, the rope at the side tends to pull the tubs off the rails.

Of all the systems of endless rope haulage the most satisfactory is the over-rope system, where the rope is carried by bearing up or bearing down mushroom pulleys and the tubs attached by lashing chains.

In the endless rope system it is essential that the rope be kept constantly tight. This may be done at any convenient point by passing the rope around a sheave carried by a tram, to which either a tension-screw or a hanging weight is attached to draw back the sheave to take up any slack rope (see fig. 176).

This system of haulage is suitable for undulating planes, as all the advantages of gravitation are utilized, and it can be worked where there are curves and branch roads. For the working of a branch road a separate endless rope is necessary, which may be driven by the main endless rope or by a separate engine or motor. To work a branch from the main endless rope the latter passes round a pulley at the branch end, the pulley being keyed to a vertical shaft, upon which shaft is fitted another pulley, called a friction wheel, which drives the branch endless rope (see fig. 177). The arrangements at the junction depend upon whether the rope is used over or under the tubs. With over ropes the roadways are arranged with gradients so that the full tubs will become detached from the rope at the junction and gravitate to the main endless rope, and the empty tubs gravitate from the main to the branch. With under ropes, the ropes at junctions are arranged to run in grooves cut below the level of the roadway to

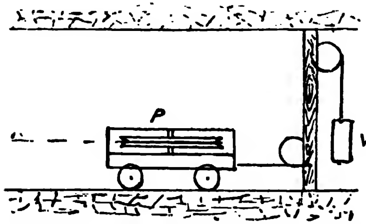


Fig. 176.—Endless Rope Tension Arrangement

R, Endless rope passing round pulley P on tram. W, Weight to keep endless rope tight.

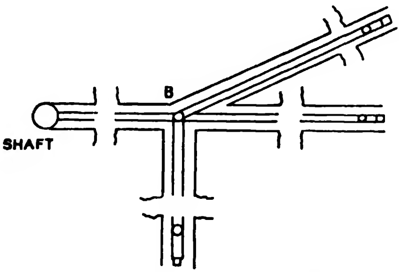


Fig. 177.—Endless Rope Haulage

Main endless from shaft actuating three branch roads through pulleys on vertical shaft B. Return wheels with tension arrangement at end of each branch.

allow the tubs, both full and empty, to cross over without coming into contact with the ropes.

A good deal of care is necessary in laying the rails at a junction where endless rope haulage is in operation, with either over or under ropes, to ensure satisfactory working, as any stoppage at such a point affects the whole of the haulage. Reliable workmen are necessary at junctions to regulate the traffic.

As the speed of the tubs is only from 2 to 4 miles per hour, the roads are not difficult to maintain. The supply of coal at the shaft is regular, as the tubs are attached to the rope at short intervals. The ropes last a long time.

EXAMPLE.—Find the horse-power to haul 350 tons of coal per day of 7 hours up an incline 750 fathoms long and rising 1 in 10. Weight of rope 5 lb. per fathom. Speed 2 miles per hour. Friction 32 lb. per ton. Tubs 4 cwt. empty and holding 10 cwt. of coal. Endless rope system.

350 tons = 700 tubs.

$\frac{700}{7}$ = 100 tubs per hour.

Speed of rope $\frac{2 \times 5280}{60}$ = 176 feet per minute.

Distance between tubs = $\frac{2 \times 5280}{100}$ = 105·6, say 105 feet.

Length of incline = 750 × 6 = 4500 feet.

Number of tubs on each side = $\frac{4500}{105}$ = 43.

Rope friction = $\frac{750 \times 2 \times 5}{20}$ = 375 lb.

$$
\begin{aligned}
W &= 43 \times 14 = 602 \text{ cwt.} = 30{\cdot}1 \text{ tons} = 67{,}424 \text{ lb.}\\
w &= 43 \times 4 = 172 \text{ cwt.} = 8{\cdot}6 \text{ tons} = 19{,}264 \text{ lb.}\\
F &= 30{\cdot}1 \times 32 = 963{\cdot}2 \text{ lb.}\\
f &= 8{\cdot}6 \times 32 = 275{\cdot}2 \text{ lb.}\\
T &= F + f + \text{rope } f + G - g\\
&= F + f + \text{rope } f + WI - wI\\
&= 963{\cdot}2 + 275{\cdot}2 + 375 + \frac{67{,}424}{10} - \frac{19{,}264}{10}\\
&= 6329{\cdot}4 \text{ lb.}
\end{aligned}
$$

Horse-power

$$= \frac{\text{tractive force in lb.} \times \text{distance moved in feet per min.}}{33{,}000}$$

$$= \frac{6329{\cdot}4 \times 176}{33{,}000} = 33{\cdot}7, \text{ say } 34.$$

Endless Chain.—In this system the arrangement is somewhat similar to the endless rope over the tubs. A double line of rails is used, one for the empties going in, the other for the full ones coming out. The driving wheel is worked by suitable gearing from the engine. To prevent the chain slipping around the wheel, forks are placed in the trod of the latter a few inches apart. The chain travels continuously in one direction at a very slow speed, and passes around an ordinary sheave at the inbye end of the engine-plane. The tubs are attached singly to the chain at intervals of from 10 to 30 yards by a very simple arrangement. In the centre of the end of each tub at the top is a fork with a groove about 1½ inches deep, and at the start the tub is pushed under the chain until the latter automatically drops into the groove (see fig. 178), and then moves the tub forward. Care should be taken that the tubs are not placed so far apart as to allow the chain to drag upon the floor. This will wear the chain, and add to the work of the engine.

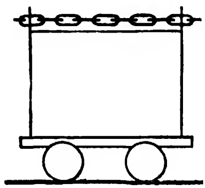


Fig. 178

This system of haulage is not much used underground now, as it is more costly and not so applicable to underground work as the endless rope.

CHAPTER XXVII

DRAWING OR WINDING ARRANGEMENTS

Temporary Winding Appliances, Ladders, Shaft Guides, Cages, Ropes, Signals, &c.

Temporary Winding Appliances.

Windlass.—For raising material from a small shaft or from an underground staple from depths of less than

15 fathoms, the windlass or *jack-roll* worked by hand is very frequently adopted. It consists of two wood uprights mortised into two bearers stretching across the shaft. Stays are fixed to the uprights on each side to support them. The barrel upon which the rope is wound is of wood, sometimes hooped with iron at the ends, and is from 4 to 6 feet in length and 10 or 12 inches diameter. Iron handles are fastened to the barrel at each end, as shown in fig. 179.

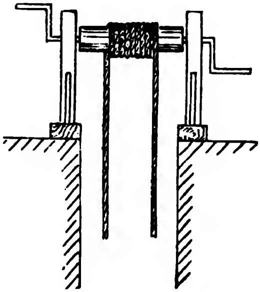


Fig. 179.—Windlass

Horse Whim or Gin.—For heavier loads and greater depths the machine sometimes used is the horse whim or gin, worked by one or two horses. It consists of a vertical wooden axis 10 or 12 inches diameter and 12 or 14 feet long, which turns in a block of stone or iron casting at the bottom end, and in an iron socket at the top supported by a long "span beam". This is made about 30 to 36 feet in length, and is supported by inclined legs at each end, which are mortised into the beam. A rope drum of from 6 to 10 feet diameter is built around the vertical axis near the top, as shown in fig. 180, and beneath it is placed the driving horizontal beam, at each end of which a horse may be attached. The ropes are conducted over little guide pulleys from the drum to pulleys fixed in the frame over the shaft. As the horses travel around with the

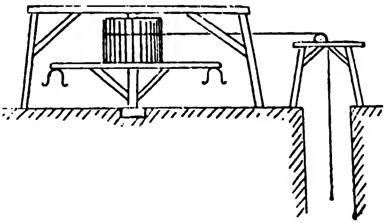


Fig. 180.—Horse Whim or Gin

driving beam, one rope is wound on the drum and the other passes off.

Ladders.—Sometimes ladders are fixed in shallow shafts for the descent and ascent of workmen. In underground staples communicating between different seams they are often used. Ladders for mining purposes are made sometimes entirely of wood, sometimes of wood with iron rungs, and sometimes entirely of iron. In the shaft, platforms are erected at intervals of from 20 to 30 feet, and between them the ladders are placed as shown in fig. 181.

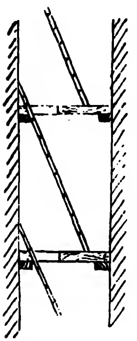


Fig. 181.—Arrangement of Iron Ladders in a Shaft

Ladders made of wire rope, with iron rungs fastened between the strands of the rope, are sometimes used in underground staples where it is difficult to convey and fix rigid ladders. Such a ladder is made generally in one length, and is stretched from top to bottom; it is securely fastened to a balk at both ends, and at such intermediate points as may be deemed necessary to prevent it swaying with men upon it.

Guides or Conductors.—When the sinking of the shaft which is intended to be used for winding is completed, guides or conductors, up and down which the cages slide, are then fixed in the shaft. Two cages are generally used, one ascending with full tubs while the other is descending with empty tubs, and by sliding up and down the guides they are prevented from coming into collision with each other or with the sides of the shaft, and are kept in a vertical line. Guides are made of wood, iron rods, iron or steel rails, or wire ropes.

Wood Guides were formerly much used. They are generally made of pitch or memel pine in lengths of 18 feet to 20 feet, and 4 inches × 3 inches section, placed

vertically down the shaft from the top to the bottom, and bolted to buntons. The actual length should, for the sake of spacing the joints, be a multiple of the distance between the buntons. The latter are cross pieces of timber fixed across the shaft and let into the sides, and the guides are secured to them by countersunk bolts. The joints of the guides are formed by the ends being bevelled so as to overlap and make the contact level. Wood guides are liable to break, and require frequent renewal.

Iron or Steel Flat Bottom Rails are much used. They are fixed to wood buntons placed at intervals of about 6 feet from the top to the bottom of the shaft. Cast-iron plates or sleepers are bolted to the buntons, and the rails are laid in grooves in the plates, and secured by bolts passing through the buntons (see fig. 182). Rail guides are very substantial and permanent, and where heavy cages are used, and a high speed of winding is attained, they give every satisfaction. They seldom need repairing or renewing, and if put in properly, give a steady motion to the cage. The first cost is more than that of any of the other kinds of guides.

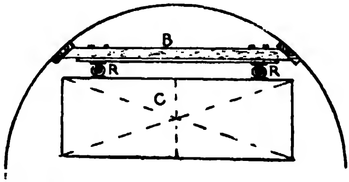


Fig. 182.—Arrangement of Rail Guides

B, Wood buntons secured to shaft sides.
R, Rails and sleepers secured to buntons.
C, Cages and shoes encircling guides.

Wire-rope Guides are also much used. The ropes are suspended in the shaft from the pit-head framing. Sometimes they are made fast at the pit bottom and attached to screws at the top, by which they are kept tight. Another method is to secure them to the frame at the top with two or more pairs of wrought-iron clamms, which are bolted so as to grip the rope very tightly and rest upon the timbers or cross-beams. At the bottom end,

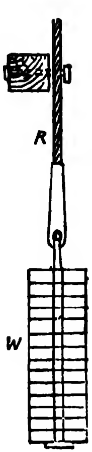


Fig. 183.—Wire-rope Guides in Shaft, showing Arrangement at Bottom for keeping Ropes rigid

R, Rope.
W, Weights.

below the level of the seam, very heavy weights are hung beneath balks of timber to keep the ropes as rigid as possible, and so prevent oscillation of the cages when running (see fig. 183). They should always be so secured as to prevent them becoming slack when there is expansion from heat, or becoming too tight when there is contraction from cold.

Wire-rope guides require no buntons in the shaft—consequently there is very little of the shaft space taken up, and the shaft sides are not interfered with. The vertical line is kept, and the movement of the cage is free and easy. The ropes should be regularly examined to discover any broken wires, or any other sign of weakness.

Number and Arrangement of Guides.—These depend upon the kind of guide used, and the size and form of cage.

When wood or iron-rod guides are used there are generally two for each cage. If the cage carries one tub only on a deck, the guides are often arranged as in

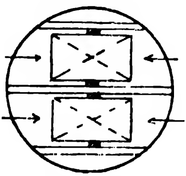


Fig. 184.—Arrangement of Wood Guides when Cage carries One Tub on a Deck

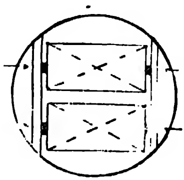


Fig. 185.—Arrangement of Wood Guides when Cage carries Two on a Deck

fig. 184; and if it carries two or more on a deck they are generally placed at the end of the cage, as in fig. 185.

When rails are used, two are generally placed on one side of each cage, as shown in fig. 186. The cages at "meetings", which is the point they pass each other in the shaft, should have a clearance of not less than 9 inches.

When wire-rope guides are used they should be arranged to give the cages as much clearance of the shaft sides and of each other as possible. There is always a certain amount of oscillation in deep shafts, where the winding is necessarily done at high speed, and there is always present a danger of the cages catching. Three guides are often used, and sometimes four, to each cage, and may be arranged as shown in figs. 187, 188.

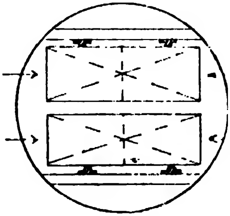


Fig. 186.—Usual Arrangement of Rail Guides. The arrows show where the tubs enter the cage

A plan is sometimes adopted of suspending two additional ropes between the cages, and attaching along the side of each cage that faces the other cage a rubbing

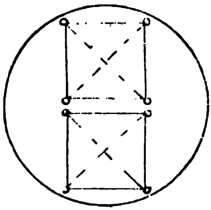


Fig. 187.—Arrangement of Wire-rope Guides. Four ropes for each cage

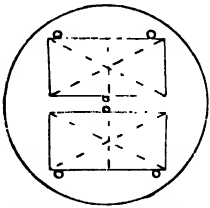


Fig. 188.—Three Wire-rope Guides

piece of timber a few inches thick; this rubs against the ropes and prevents the cages striking each other when passing. This method is more satisfactory than leaving a wide clearance, and when adopted the space between the cages need not be more than a few inches,

Cages.—These are the vehicles for conveying the empty tubs into the mine and raising the full ones to the surface. They are also the usual medium by which the raising and lowering of all workmen, tools, materials, and frequently horses and ponies, is accomplished (see fig. 189).

Cages are made of wrought iron or steel. Their form and dimensions depend upon the size of the shaft in which they have to work. A cage to hold one tub is of nearly square form, and to hold two on one deck it is rectangular. Cages are made with from one to four decks, and carry one, two, or four tubs on each deck as arranged. Rails are laid on the floor of each deck upon which the tubs stand in the cage, and the tubs are kept fixed during the motion of the cage by means of a lever, called a *catch* or *sneck*, placed at each end of the cage at each deck to catch the top or the axle of the tub. Each cage is fitted with slides or shoes of bell-mouthed shape, which fit loosely to the three sides of rigid guides, but pass around wire-rope guides. Cages are suspended from the ropes by four or six wrought-iron chains. In some instances an adjustable screw is placed on each chain to equalize their length, in order that the cage may hang evenly. Cage chains should be annealed at least once every three months.

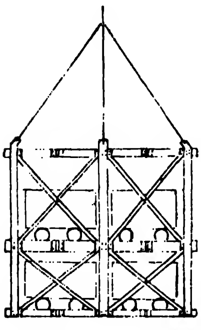


Fig. 189.—Double-decked Cage; two tubs on each deck

Ropes.—For winding cages, round or flat ropes made of iron or steel wire are chiefly used. Round and flat ropes of hemp or of aloe-fibre were at one time much used, especially flat ones, but they are seldom used now. Round hemp ropes are still used in sinking and pumping shafts for supporting or lifting heavy weights such as pumps. Galvanized wire ropes are sometimes used in

wet and upcast shafts. Round ropes made of steel wire are generally preferred for winding purposes. Wire-ropes may be divided into the following classes, the basis being the construction of the rope:

1. Ordinary strand.
2. Lang's lay.
3. Locked coil.
4. Flattened strand.
5. Flat ropes.
6. Tapered ropes.

For winding, classes 2, 3, and 4 are those generally preferred. Lang's lay wears well due to its construction. The locked coil presents a smooth surface and is extremely light for its strength; it is not quite so flexible as the other types, although when compared strength for strength it is not far behind. The flattened strand type has little internal friction, is non-spinning, and exceedingly flexible. These ropes are made from (*a*) charcoal iron, (*b*) crucible steel, and (*c*) high-tensile or plough steel, the strength of these materials varying from about 40 tons per square inch in charcoal iron to over 120 tons per square inch in high-tensile steel.

The wires forming the rope are selected after test specimens have shown their tensile strength, and the following empirical formula connecting the size and strength of the finished rope has been obtained.

STRENGTH OF ROUND ROPES

Let W = Breaking or proof load in tons.
,, C = Circumference of the rope in inches.

Hemp rope .. $W = 0{\cdot}25\, C^2 \therefore C = \sqrt{\dfrac{W}{0{\cdot}25}}$.

Iron-wire rope $W = 1{\cdot}50\, C^2 \therefore C = \sqrt{\dfrac{W}{1{\cdot}50}}$.

Steel-wire rope $W = 3{\cdot}00\, C^2 \therefore C = \sqrt{\dfrac{W}{3{\cdot}00}}$.

Plough steel rope $W = 4{\cdot}00\, C^2 \therefore C = \sqrt{\dfrac{W}{4{\cdot}00}}$.

The safety load of round ropes, i.e. the heaviest weight to be placed upon a rope in working, may be taken at from $\frac{1}{6}$ to $\frac{1}{8}$ of the breaking load. For example, if the breaking load is 36 tons, the working load of the rope should not exceed 6 tons. With winding ropes the safe or working load is usually taken as $\frac{1}{10}$ of the breaking load.

In calculating the weight upon a rope in a deep shaft the weight of the rope itself should not be forgotten.

The weight of round ropes may be calculated approximately by the following formula:

Let C = Circumference of rope in inches.
" W = Weight in pounds per fathom.

Hemp rope .. .. $W = \frac{C^2}{4}$.

Iron or steel wire .. $W = C^2$.

EXAMPLE.—Find the size and weight per fathom of a plough steel winding rope to raise a load of 5 tons, exclusive of its own weight, from a depth of 200 fathoms, the safety factor being 10.

Weight of rope in tons $= \frac{200\ C^2}{2240}$,

then $C^2 \times 4$ = working load × safety factor + weight of rope × safety factor

$$= 5 \times 10 + \left(\frac{200\ C^2}{2240}\right)10.$$

$$= 50 + 0{\cdot}9\ C^2;$$

$$\therefore 3{\cdot}1\ C^2 = 50,$$

$$\text{and } C^2 = \frac{50}{3{\cdot}1} = 16.$$

$$C = \sqrt{16} = 4.$$

Rope is 4 inches circumference or $\frac{4}{3{\cdot}14} = 1\frac{1}{4}$ inches diameter, and weighs 16 lb. per fathom or a total weight of approximately 1·43 tons.

Strength of Flat Ropes.—Flat ropes are formed of two or more round ropes stitched together. The strength of one is calculated, and the result, multiplied by the number of round ropes forming the flat one, gives the breaking strength. From $\frac{1}{8}$ to $\frac{1}{10}$ of the breaking load may

be taken as the working load. C. M. Percy, in his *Mechanical Engineering of Collieries*, gives the following rules:

FLAT ROPES

Width in inches × Thickness in inches × 35 Charcoal Iron = Safe working load in cwt.
Width in inches × Thickness in inches × 55 Crucible Steel = Safe working load in cwt.
Width in inches × Thickness in inches × 70 Plough Steel = Safe working load in cwt.

These flat ropes are rarely used as, owing to difficulties of construction, they are more expensive to manufacture than round ropes, and weight for weight do not give the same strength.

Taper ropes are used to a limited extent in deep winding. Their cross-section is varied between the cage and the drum end, so that the safety factor will be constant throughout their length. The method of construction most in favour is to use the same number of wires throughout the length of the rope, but altering their diameter from point to point. Modern methods of arc welding make this quite possible without seriously interfering with the strength of the wires at the weld.

Duration, Examination, and Protection of Ropes.—The life of a rope greatly depends upon the quality of its manufacture and the circumstances under which it works. If the shaft is wet and the water of an acid nature, or the shaft is an upcast with a furnace at the bottom, the rope may wear out quickly. But in general the number and speed of the windings have the greatest effect on the duration of a rope. The average duration is 12 to 18 months, but the work done is more important than the number of months. Under the 1911 Mines Act, the winding rope must be renewed at intervals of not more than three years.

The examination of winding ropes forms part of the daily inspection at a colliery. They are slowly uncoiled from the drums, and passed through the hand of the

examiner, who feels and observes particularly for any signs of wear, broken wires, or other irregularity. If any broken wires are found the rope should be immediately condemned.

All winding ropes should be periodically oiled with thick oil, to protect them from the atmosphere and other destructive agents and prevent corrosion. Oil requires to be more frequently applied in wet than in dry shafts.

To reduce the violent strain on winding ropes in lifting the loaded cage from the bottom of deep shafts, springs are sometimes placed under the pulley carriages, and sometimes at the bottom of the shaft for the cage to rest upon. To get the best life out of winding ropes they should not be worked round drums or pulleys of less than 100 times their own diameter.

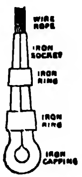


Fig. 190.—Wire-rope Socket or Capping to connect Rope and Cage Chains

Attachment of Rope to Cage Chains.—The cage chains all terminate at the top in one large link, to which is attached a D link. Above this is placed a detaching hook, and the end of the winding rope is connected by an iron socket or capping. No method of capping shall be used which fails to withstand a strain of at least seven times the weight of the maximum load (fig. 190).

Solid-made sockets, afterwards bored out, are much used. The rope is fixed by running in white metal. Another method is to have a socket constructed to form a cone, the rope being enclosed by a pair of semicircular wedges, which, when together, form a cone having the same taper as the socket. The rope-ends are untwisted and bent over the cone, and inserted in the socket. The hoops are then driven on tightly. This forms a strong capping when carefully done.

Keps.—These are sometimes known as *fans*, or *shuts*, and are a kind of skeleton platform, worked by an

arrangement of counterbalanced levers, for the cage to rest upon at the top and bottom of the shaft during the changing of the tubs. Those on the surface when in position are pressed back against the shaft side by the ascending cage, which is then raised a few inches above them. They then fall again into position to receive the cage, which is lowered on to them. When the tubs are changed the cage is slightly raised, and the keps are drawn back by a lever to allow the cage to descend the shaft. At the shaft bottom keps are used only when the cage has two or more decks. In order to avoid having to lift the cage before descent, patent keps are now much used. When the tubs are changed and the cage is ready for descent, a hand lever is pulled over, which liberates the keps in such a way that the weight of the cage assists to push them clear to allow of the descent of the latter.

Shaft Signals.—Arrangements must be made in every shaft for the communication of signals between the shaft top, bottom, and every intermediate working level and the engine, and vice versa, so that the engine-man may start and stop the cages as required by those in charge at the shaft. The code of signals specified by the Regulations must be placed at every stopping-place.

A common signalling arrangement is to have a small iron-wire rope with a lever at one end and a hammer at the other, which is pulled to give the required number of raps. A bell is sometimes used instead of a hammer.

Electric signals are also used, especially for deep mines, as they are much quicker and more reliable. Telephones are also often used in shafts and underground. The Mines Act states that "every winding engine shall be provided with an appliance which shall automatically and visibly indicate the nature of the signal until the singal is complied with ". In order to comply with this part of the Act, a large number of so-called visual indicators have been devised. They consist generally of some arrangement of mechanism that moves a pointer on a figured dial, the hand standing at the signal indi-

cated. As soon as the signal is acted on and the winding commenced, the engine trips out the signal and leaves the indicator clear. A very large number of these visual indicators are in use, and it would be unfair to select any specific type for special mention, as there is little to choose between them.

CHAPTER XXVIII

DRAWING OR WINDING ARRANGEMENTS—CONTINUED

Construction of Pit-head Frames. Pulleys. Appliances for the Prevention of Overwinding

Pit-head Frames.—At the top of the winding shaft, pit-head or pulley frames, as they are often termed, are erected to carry the pulleys or sheaves over which the ropes pass into the shaft. They are constructed of either wood or iron. Wood pulley frames have been very much adopted. They consist mainly of two or four upright posts or pulley legs, and two back stays set at an angle from the top of the pulley legs. The pulley legs support the vertical strain of the load, and the back stays resist the pull towards the engine. An arrangement of pulley frame frequently adopted is shown in fig. 191. The height of the pulley frames depends upon the height of the staging where the tubs are changed, and upon the height required above that for the convenient and safe working of the cage. The height varies from 30 to 80 feet, the sectional size being in proportion to the height. The back stays should be of the same sectional size as the legs, and both should rest upon masonry pillars. The pulley legs should be parallel to the vertical line of the ropes in the shaft, and the back stays as nearly as possible parallel to the line of the rope between drum and pulley.

Steel pit-head frames are largely adopted at new mines. At large mines newly opened, steel frames are made compulsory by the Coal Mines Act, 1911, in order to prevent the possibility of fires. The general arrangement is similar to that of wood frames. The legs are usually of open lattice-work, with angle-iron bars connected by flat bars, and braced together by plate girders and spandrils.

A *pulley* or sheave for each rope is placed at the top of the legs, to suit the lead of the ropes off the drum and the line of the ropes in the shaft. The rims and central boss of the pulleys are made of cast iron, and the arms of wrought iron. The rims are grooved circular to fit round ropes, and flat for flat ropes.

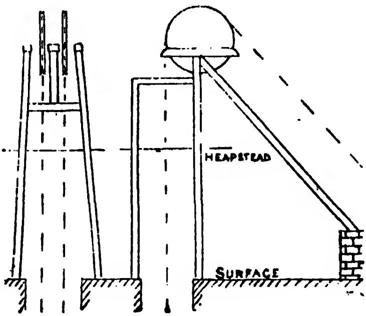


Fig. 191.—Side and End View of Pulley Frames and Pulleys

Pulleys vary in diameter from 10 feet to 20 feet. It is a good plan to have the diameter of the pulleys made about the same as that of the drums, and both should be made as large as the engine will permit of. The following rule has been given for the diameter of round iron or steel rope pulleys, viz.:

Rope,	1 inch	circumference,	requires pulley	10 ft.	diameter,
,,	$1\frac{1}{4}$,,	,,	,,	$10\frac{1}{2}$ ft.	,,
,,	$1\frac{1}{2}$,,	,,	,,	11 ft.	,,
,,	$1\frac{3}{4}$,,	,,	,,	$11\frac{1}{2}$ ft.	,,

and so on.

Appliances for the Prevention of Overwinding. —The Coal Mines Act, 1911, stipulates that "where the apparatus ordinarily used for raising or lowering

persons to or from the surface is worked by mechanical power it shall, if the shaft is vertical, be provided with a detaching hook, and, if the shaft is more than 100 yards in depth, shall also be provided with an effective automatic contrivance to prevent overwinding ".

Overwinding occurs when the cage in ascending is not stopped at the landing-stage, but dashes up to the pulleys, with the result that the rope breaks and the cage falls down the shaft, unless some of the following appliances are in use.

Detaching Hooks.—These do not exactly prevent overwinding, but they avoid the evil consequences. They come into action before the cage reaches the pulleys, releasing the rope, which passes freely away, and the cage is prevented from falling down the shaft by the detaching hook, which catches upon a cross beam or girder fixed a short distance below the pulleys. The detaching hooks not only detach the rope from the cage, but suspend the cage and its occupants; they have thus been the means on many occasions of saving lives and property.

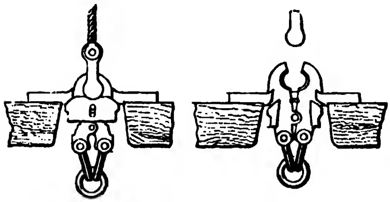


Fig. 192

Walker's Hook.—This hook is an invention of Mr. William Walker. The supporting ring through which the rope constantly works, and upon which the hook catches and hangs after detachment, is a fixture in a balk of timber or iron girder a short distance below the pulleys. The hook consists of a pair of jaws working on a centre pin, and so arranged that the weight of the cage has a tendency to open them at the top, but this is prevented by means of a clamp which passes around them and is kept in position by a pair of copper pins (see fig. 192). In case of overwinding the jaw hooks

pass freely into the supporting ring, but the clamp, coming in contact with the bottom flange of the ring, is held stationary. The jaw hooks are thus released from the clamp, and, as soon as they reach the top of the ring, are forced open by the weight of the cage and hook themselves on the top of the ring. The rope passes away over the pulley, while the cage is safely suspended.

Ormerod's Hook.—This was invented by Mr. Edward Ormerod of Manchester, and belongs to the class of plate hooks. It is formed of three plates, and when in working order is wider at the bottom than at the top; but in the event of overwinding, the hook is drawn into the bell-mouthed cylinder in the suspending balk or girder and the wide parts of the hook, coming in contact with the inner side of the cylinder, are pressed inwards, while the top parts are pressed outwards and the projections or hooks catch over the top. At the same time the rope is released, and the bottom shackle drops into the slot and locks the hook firmly in its position, and safely suspends the cage. With this hook an arrangement is made to facilitate the lowering of the cage after detachment (see fig. 193).

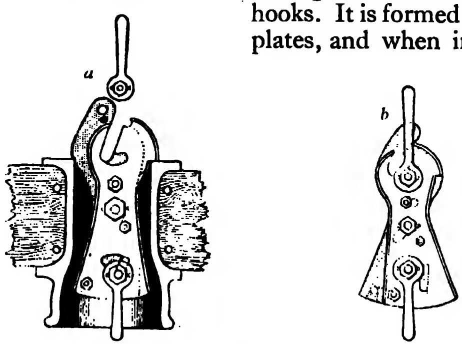


Fig. 193.—The Ormerod Safety Link

a represents the position it assumes when wound up into the cylinder, the rope shackle being disconnected and the link firmly locked in position. *b* is a front view of the safety link when in working order.

There are several other detaching hooks made and used, but it should not be forgotten that unless the cage is moving at a low rate of speed when it strikes the cross beams, the mere freeing of the rope by the detaching hook is not sufficient protection, as the energy stored up in the moving mass has to be arrested instantaneously, and this cannot be done at high speed without damage to both cage and head-gear.

Overwind Controller.—This is an automatic contrivance attached to the winding engine which comes into operation in the event of the engine (*a*) attaining from any cause an excessive speed during any part of the wind, (*b*) failing to reduce speed near the pit top, and (*c*) starting in the wrong direction. It shuts off the steam and applies the brakes in a gradually increasing manner, and brings the cages to a stop in a very short distance. The ascending cage is stopped from being overwound, and the descending cage from being dashed into the sump. It is important, not merely to have a sensitive and efficient controller to come into operation when necessary, but the brakes on the winding drum should be of sufficient strength and reliability.

Appliances to Prevent the Cage Falling when the Rope Breaks.

Safety Cages.—Sometimes cages are fitted with an arrangement for arresting their descent when the rope breaks. Such contrivances are very seldom used, because generally they cause much trouble by coming into action during rapid winding, and frequently fail when they ought to act. Most of them are arranged for use with wood guides, which they are expected to grip with sufficient force to stop the cage immediately it begins to descend faster than in the ordinary course of winding.

Broadbent's Patent Safety Cage has a pair of eccentrics, which are placed one on each side of the guides, but are kept from pressing upon the latter during winding by suspension chains. During winding the cage chains and suspension chains are tight, but when they slacken or

drop, as in the case of the rope breaking, a spring forces the eccentrics against the guides with sufficient pressure to arrest the descent of the falling cage. This apparatus may be made to act upon wire-rope guides.

CHAPTER XXIX

DRAWING OR WINDING ARRANGEMENTS—CONTINUED

Winding Engines. Drums for Flat and Round Ropes. Spiral Drums, Balance Chains, and other Methods for Equalizing Load on Engine.

Winding Engines.—These may be beam, geared, vertical, or horizontal engines.

Beam Engines at one time were much used, but are rarely used now for winding purposes, owing to the unnecessary weight of the beam and to its liability to fracture.

Geared Engines are sometimes used at small mines, where they can be applied for winding during the day and for pumping during the night. With geared engines a brake must be applied to the drum.

Vertical Engines, direct acting, with single or double cylinder, are in use at many mines for winding, and usually give good results.

Horizontal Engines, direct acting, with double cylinders, are usually considered to be the best type of engines for winding (see fig. 194). They have this great advantage that all the working parts are visible to the engineman working the engine. Horizontal engines with one cylinder only should not be used for winding, owing to the difficulty of getting " over the centre " when moving slowly, or of getting started when the engine has been stopped " on the centre ".

As a rule winding engines work *unexpansively*—although

the cut-off can be varied by means of the link motion with which they are usually fitted—and are *non-condensing*, because the irregular, intermittent, and rapid nature of winding are conditions unfavourable for the general application of expansion and condensation. At a few places, expansion gear, with special arrangement to suit the conditions of winding, has been applied to the engines. Compound winding engines have been adopted in a few instances and work satisfactorily with economical results.

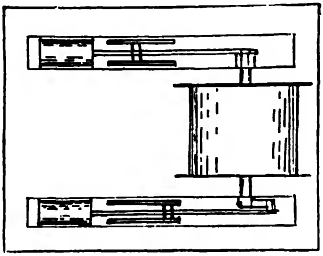


Fig. 194.—Horizontal Winding Engine

Drums.

Flat-rope Drum.—For flat ropes the vertical drum is used. It consists of a small barrel with vertical arms of wood fixed upon the drum shaft of the winding engine. The arms are placed apart at such distance as will allow the flat rope to work between them without friction. Very little clearance is required, or there is a danger of the rope slipping (see fig. 195). Cast-iron vertical drums, dispensing with the wood arms and having proper cast-iron flanges all around the drum, have in some instances been adopted.

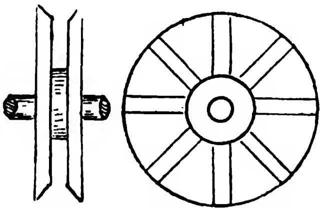


Fig. 195.—End and Side View of Drum for Flat Rope

Round-rope Drums.—For round ropes a cylindrical drum is frequently used. It consists of cast-iron rings, which form the actual drum or barrel, and to its sides are bolted cast-iron or steel flanges to prevent the rope

getting off. In some cases wood lags are bolted to the drum to form a wood surface for the ropes to coil upon. One drum serves for both of the winding ropes, one of which coils over the top of the drum and the other under it, the drum being made of sufficient width to allow of this (see figs. 196, 197). The cylindrical drum is universally used for haulage.

The Spiral Drum.—This is for round ropes, and is sometimes called the *conical* or *scroll drum.* It is made

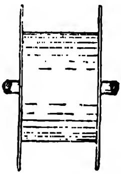


Fig. 196.—Cylindrical Drum for Round Ropes

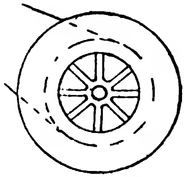


Fig. 197.—Side View of Cylindrical Drum

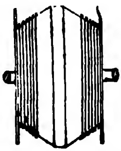


Fig. 198.—Spiral Drum for equalizing Load on Engine

of two diameters, and the barrel is thus inclined or conical, as shown in fig. 198. Thus, while the rope on one side is being wound on coils upon an increasing diameter, the one on the opposite side coils off a gradually decreasing diameter. At Boldon Colliery, in the county of Durham, there is a conical drum 30 feet in the centre at its greatest diameter, and 19 feet at the sides where its diameter is least.

Spiral drums are considered to be the best, but they are very unwieldy, and a heavy dead weight to start and stop at each winding. Sometimes they are plain and ungrooved, but they are much safer when formed with grooves for the rope to work in.

Methods for Equalizing Load on Engine.

Spiral Drum.—The object of the spiral drum is to counterbalance or equalize the load upon the engine. When a plain cylindrical drum is used, and unbalanced, the lift is very unequal. The greatest strain upon the

engine is at the start, with the cage of full tubs at the bottom of the shaft and the full length of rope attached to it, and the least strain is when the full cage reaches the surface.

In the spiral drum the diameters are proportioned, and the strain upon the engine is uniform from the beginning to the end of each winding. The diameter of the drum is least when it commences to raise the full cage from the bottom, and greatest on the side of the descending cage. The former thus has the least leverage with the greatest weight, and the latter the greatest leverage with the least weight. These conditions alter during the process of winding, with the result that the work of the engine is balanced at every part of it.

EXAMPLE.—Find the dimensions of a spiral drum to lift the following load from a depth of 200 fathoms and provide a balanced load upon the engine throughout the run.

Load of coal	25 cwt.
2 hutches	10 „
Cage	15 „
Rope	30 „

Taking moments at the lift, meetings, and landing for both cages we get:

	Load Side.	Empty Side.	Resultant.
Lift	+ 80	− 25	+ 55
Meetings	+ 65	− 40	+ 25
Landing	+ 50	− 55	− 5

The minimum diameter of drum is fixed by the diameter of the winding rope, but taking it as 12 feet and maximum as D feet, then by the principle of moments:

$$(80 \times 12) - 25\,D = 50\,D - (12 \times 55)$$
$$960 - 25\,D = 50\,D - 660$$
$$-75\,D = -960 - 660;$$
$$\therefore D = \frac{1620}{75} = 21{\cdot}6 \text{ feet.}$$

So that a perfect balance with the given load will be obtained

throughout the wind when the drum has a minimum diameter of 12 feet and a maximum of 21·6 feet.

Chain and Staple.—In this counterbalance arrangement a chain coils upon a small drum keyed upon the main drum shaft, and uncoils into a staple or small shaft usually placed behind the engine (see fig. 199). To the end of the small drum chain heavy chains are attached. When the cages are at the top and bottom of the shaft these chains are hanging in the staple. During the process of winding they are lowered to the bottom

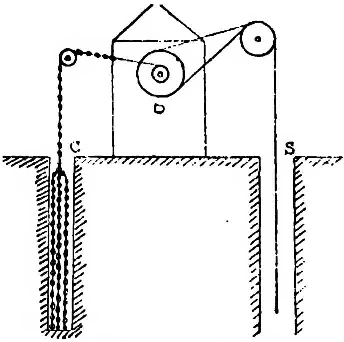


Fig. 199.—Chain and Staple Method of Counterbalancing

C, Chain staple. S, Winding shaft. D. Winding drum and small drums for chain.

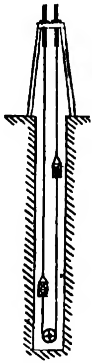


Fig. 200. — Tail Rope under the Cages counterbalance

of the staple, and are completely at the bottom when the cages are at " meetings ". As the cages pass each other the chain upon the small drum winds in the opposite direction, and gradually raises the heavy chains until, when the winding is finished, they are in the same position as at the start. Up to the point of meetings, the ascent of the full cage is thus assisted by the heavy chains, thereafter it is retarded.

Tail Rope under Cages.—This is another method of counterbalancing, and is applied with a plain cylindrical

drum. It consists of a rope passing round a pulley placed near the pit bottom, and attached to the bottom of each cage (see fig. 200). The ascending cage draws the rope round the pulley and up the shaft, thus rendering the weight of rope upon the engine always about uniform. The engine is thus enabled to start the cages more easily, and perform the winding quicker. It should be noted that the strength of the winding ropes should be increased in proportion to the weight of the tail rope when drawn up to the surface by the ascending cage.

Brakes for Winding Engines.

Foot Brake.—In winding engines of ordinary size, a brake must be placed upon the drum to be worked by pressure from the foot of the engineman through an arrangement of levers. The brake consists of an iron strap 6 or 8 inches in width, which sometimes passes half-way and sometimes the whole of the way around the drum. The ring of the drum upon which the brake acts is generally cleaded with segments of oak to make the brake grip better. Brakes on the drums are much better than those on fly-wheels, because, in the event of any breakage of machinery, the brake on the drum may be applied to prevent the cages sliding down the shaft too quickly.

Steam Brakes.—These are applied by pressure of steam in a small steam cylinder. The valve for admitting the steam into the cylinder when the brake is required is worked by the engineman by means of a lever. Where large powerful engines with drums of large diameter are used for winding, steam brakes should be adopted, so that the engines may be well under the control of the engineman.

Winding Indicator.—A winding indicator should be placed in front of the engineman at every engine, so that he may see the exact position of the cages during the whole process of winding. The indicator generally consists of a vertical frame or miniature shaft, up and

down which a weight slides, moved by a small chain or cord which is wound upon the drum shaft. When the cage is at the surface the weight is at the top of the frame, and descends with the cage; at "meetings" it begins to ascend, and when nearing the top is made to ring a bell as a warning to the engineman that the ascending cage is near the surface. The indicator is usually marked in such a way as to denote various points in the shaft.

A *dial indicator* is sometimes used, with an index pointer which revolves once during one winding, and in revolving indicates the position of the cages in the shaft by well-defined marks on the dial. This indicator is worked by gearing off the main shaft.

Koepe System.—In this system drums are dispensed with entirely and a pulley substituted. One winding rope is used instead of two as in the ordinary method. This rope passes about half round an ordinarily constructed V pulley, which is fixed on the crank shaft of the winding engine, and over the pulleys at the shaft top and the ends connected to the cages.

Usually a balance rope under the cages is used in connection with this system of winding. This system does not appear likely to become largely adopted.

CHAPTER XXX

THE DRAINAGE OF MINES

Inflow of Water into Mines. Methods of Drainage: Adit-levels, Winding Water, Pumping, Lifting Sett

Water in Mines.—There are few situations where strata are passed through by vertical bore-holes and shafts but are found to contain water in greater or less quantities, according to the nature of the rocks. The rock-beds may be divided into the pervious, those

through which water percolates easily, and the impervious, those which resist the passage of water. Limestone and sandstone are examples of pervious rocks, and clay and shale are examples of impervious rocks. The water contained in the pores, joints, fissures, faults, and cavities of rocks has all found its way in from the surface. When rain falls on the surface a large quantity of it soaks into the ground; some of this finds its way into the pervious rocks at their outcrop, and passes along until the rocks are saturated, or until intercepted by faults and fissures.

In sinking a shaft it is usual to pass through both pervious and impervious beds, and the pervious or water-bearing beds are usually tubbed off to prevent the influx of the water into the shaft. If the coal occurs immediately under impervious rocks, the working of the seam may be carried on without much interruption from water, unless faults are met with or a heavy fall of the roof occurs, forming a break through the impervious beds to those containing the water, in which case an influx or a " feeder " of water will probably enter into the workings.

Water in passing through rocks takes up large quantities of the soluble matter which they contain. Sometimes this matter has a corroding effect upon the pumping appliances. This injurious effect is sometimes neutralized by mixing in the sump or standage some other substance before the water has to pass through the pumps. Sometimes the pumps are lined with a thin casing of wood, or more usually gun-metal, to prevent the acid water eating into them.

METHODS OF DRAINING MINES

Adit-level.—An adit or adit-level is a tunnel driven in at the foot of a hill, or the slope of a valley, to drain off the water as the mine workings advance under the rising ground. A slight inclination, just sufficient to

cause the water to flow out, is given to the level (see fig. 201). In metal-mining districts very extensive areas have been drained by adits to very considerable depths, but it very rarely happens that coal-seams are so fortunately placed that they can be drained in this way; they frequently lie at such depths below the general level of the country that adits are quite inadmissible, and the water must necessarily be lifted to the surface in order to get it removed. This is accomplished at the shafts, to which the water in the workings must be directed, and as the "feeders" run continuously, sumps and standage for their accumulation before being raised to the surface are usually made in the stone at the shaft bottom.

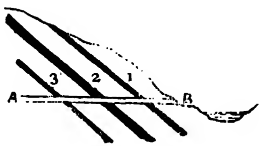


Fig. 201.—A B, Adit-level draining the coal seams 1, 2, 3, and other beds above it. The water flows from A to B.

Winding Water.—When the quantity of water to be raised is small and the mine shallow, it may be considered better to wind the water than go to the expense of pumping appliances. Where there is sufficient standage the water may be allowed to accumulate during the working hours of the day, and be raised at night.

In sinking shafts the kibble is usually employed in winding the water when the quantity is small. If the water is deep enough at the bottom, the kibble is allowed to dip and fill itself; if not, men fill it with buckets. When it reaches the surface, it is raised slightly above the level, then a tram is pushed over the shaft and the kibble lowered upon it. It is then disconnected from the chain, and an empty one connected and lowered into the shaft while the full one is being emptied, or it is emptied direct and returned into the shaft.

When water has to be raised in a winding shaft, a water tub of wood or iron upon wheels is made to fit into the cages. It is provided with a clack or valve opening upwards, placed in the bottom for the entrance of the

water as the cage dips into the sump; and at one of its ends or sides there is another clack, which is opened by a self-acting arrangement for the discharge of the water when the cage arrives at the surface.

Sometimes a tank is placed underneath the cage. It is fastened to the cage bottom by plates running the whole length or width of the cage, and is usually made of iron (see fig. 202).

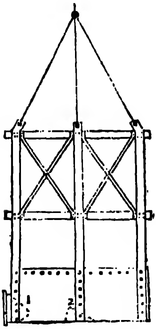


Fig. 202. — Iron Tank attached under Cage for winding water. Dotted lines, 1, show outlet valve open; and 2 show inlet valve also open.

Sometimes a shaft is divided into two compartments, one for drawing coals and the other for drawing water. In such cases a separate engine is used for the water, which is drawn in two iron tanks running in guides. A large quantity of water may be raised in this way.

The winding of water damages the shaft, ropes, guides, &c., owing to the leakage, and the violent strains caused through dipping into the sump and starting the cages or tanks. Pumping appliances are therefore more generally adopted for raising water than winding, being more convenient, efficient, and economical.

Pumping Water from Mines.—There are several kinds of pumps used in this work, viz.:

1. The Lifting Sett, with the engine at the surface.
2. The Forcing Sett, with the engine either at the surface or underground.
3. The Turbine Pump, where the motor is placed underground and coupled direct to the pump.

The Lifting Sett.—This consists of windbore, clack-piece with clack or valve, working barrel, bucket and bucket-piece, pumps or stocks, and rods or spears (see figs. 203 to 206).

The *windbore* (*a* in fig. 203) is a cast-iron pipe closed at the bottom end, with its circumference perforated

with *snore* holes for the admission of water for a distance of 3 feet to 4 feet up. It is usually made *egg-ended*, and is placed in the sump at the bottom of the sett.

The *clack-piece*, *b*, is a specially made cast-iron pipe in which the clack or valve is seated; it is made with a door—secured by bolts—through which the clack may be changed.

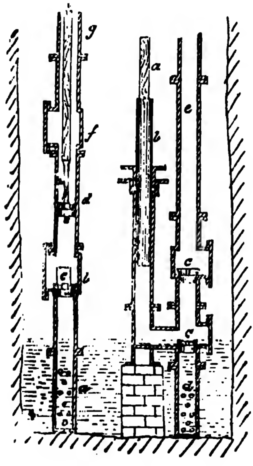


Fig. 203 Fig. 204

Fig. 203.—Main Working Parts of a Lifting Sett. Fig. 204.—Main Working Parts at the Bottom of a Plunger Sett.

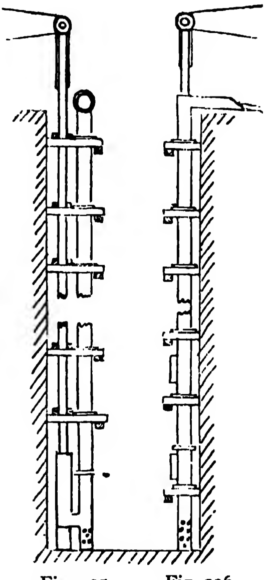


Fig. 205 Fig. 206

Fig. 205.—Arrangement of Work in Shaft of a Plunger Sett. Fig. 206.—Arrangement of a Lifting Sett showing Buntons and Collarings.

The *clack*, *c*, is made of brass or iron, jacketed with leather or gutta-percha, and having the ordinary butterfly valves of wrought-iron plates and leather. In some cases the leather wears quickly and requires frequent changing. Other forms of valves have been tried, viz., the single beat or mitre valve, the india-rubber disc

valve, and the Cornish double-beat or equilibrium valve.

The *working barrel, d*, in which the bucket works up and down, is usually bolted to the top of the clack-piece. It is made of cast-iron, and is sometimes lined with brass or gun-metal to increase its durability and reduce friction. Its length depends upon the length of the stroke given to the bucket.

The *bucket, e*, consists of an iron shell and hoop, around which is placed an india-rubber or gutta-percha jacket to fit the working barrel. The valves or falls are similar to those of the clack, and consist of iron plates grathed or faced with leather, working on a hinge in the centre and at the top of the bucket. The jacket surrounding the bucket and the falls are subject to much wear and tear; the length of time they last depends upon the state of the working barrel, the water, and the speed or number of strokes per minute.

The *bucket-piece, f*, is fixed at the top of the working barrel, and is fitted with a door called the *bucket-door*, by which access is gained to the bucket.

Pumps or Stocks, g.—These are placed upon the bucket-piece and carried up to the surface. They are generally 9 feet in length, and in all cases should be about 1 inch larger in diameter than the working barrel. They are made with thick flanges, strengthened with brackets, and with an iron belt 3 feet from each end. The joints have spigots about an inch long, and are also properly faced: upon the facings are placed wrought-iron rings ¼ inch thick, wrapped in flannel and steeped in tar; and the whole joint is secured with bolts. At the top of the pumps a " delivery-box " or " collar launder " is fixed, from which the stream of water is delivered.

Spears, h.—These are wood rods which are carried down inside the pumps from the surface to the working barrel, connecting the engine with the bucket. They are generally made of pitch-pine, in pieces of from 30 feet to 45 feet in length, the sectional size being according to the diameter of the sett and the depth. The different

spears are joined together by means of wrought-iron plates. The ends of the spears are made to fit closely together, and the plates are secured on opposite sides by bolts passing through the wood. Wood cleats are fastened to the spears at intervals to prevent the bolt-heads wearing against the sides of the pumps when working.

Pit Work Arrangements.—The lifting sett is fixed into the shaft by commencing with the windbore. This may be placed upon the bottom of the sump, or on a platform of strong balks or iron girders in the sump. The other pieces and the pumps, as already enumerated, are added above in succession until the surface is reached, care being taken to keep the whole truly vertical. The height from the water in the sump to the bucket, or the *suction* as it is termed, should not exceed 25 feet. To keep the lifting sett steady and in its vertical position, buntons are fixed across the shaft, and cross pieces of timber termed *collarings* or *horse-trees* are secured to these against the pumps (fig. 207). Scaffolds should be placed at the clack and bucket doors for the workmen to stand upon when renewing the clack and bucket.

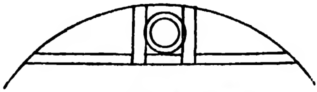


Fig. 207.—Method for securing Pumps in Shafts

That part of the sett below the bucket is termed the *suction*, and above is termed the *lift*. The height of a lift may be anything up to 50 or 60 fathoms. In some cases a lifting sett is used to a depth of 70 fathoms, but this is unusual. When the shaft reaches this or a greater depth it is customary to have two or more lifts, or lifting and forcing setts combined. The height of the lift is limited by the weight, that is, of the water in the pumps, the spears, and the bucket. In a lifting sett an endeavour is usually made to counterbalance the spears.

Action of the Lifting Sett.—The action of the pump depends upon atmospheric pressure. Theoreti-

cally, reckoning an atmospheric pressure of 15 lb. per square inch, water will rise in a perfect vacuum to a height of 34 feet. But in ordinary working pumps, owing to the imperfect vacuum obtained in practice, it will not rise more than 28 feet, and, to work well, the height of the suction should not exceed at the most 25 feet.

When the bucket makes its upstroke, its " falls " are closed by the pressure of the air above them, and a partial vacuum is thus formed. Consequently the atmospheric pressure on the surface of the water in the sump forces a column of water up through the clack, the falls of which open upwards into the working barrel. When the bucket makes its downstroke, the falls of the clack close and prevent the escape of the water, which thus passes through the falls of the bucket, which are opened by the pressure of the water below them. When the bucket takes its next upstroke it lifts this water with it, and a vacuum again being formed below, another column of water follows the bucket in its ascent. When the pumps have been filled to the top in this manner, every upstroke of the bucket gives a delivery of water at the surface, the quantity depending upon the length of the stroke and the size of the pump. In the action of the lifting pump there is thus a process of " suction " depending upon atmospheric pressure, and of " lifting " depending upon the strength of engine.

Sinking Shafts.—Water in sinking shafts is generally dealt with by a lifting sett which is lowered as the sinking progresses. After a certain depth has been attained, a forcing or plunger sett is put in, to which the lifting sett pumps. In modern practice turbine pumps are often used owing to the ease with which they can be raised and lowered in the shaft.

CHAPTER XXXI

THE DRAINAGE OF MINES—CONTINUED

Forcing or Plunger Sett of Pumps. Underground Forcing Pumps

Forcing Sett with Engine on Surface.—The forcing or plunger sett differs from the lifting sett, in that the rods or spears pass down the shaft on the outside of the pumps, and work through a stuffing-box into a working barrel. They force the water through clacks up the pumps to the surface by their downward stroke.

The forcing sett consists of spears, termed *dry* spears, because they work on the outside of the pumps, attached to the bottom of which is an iron plunger or ram, a working barrel or plunger case with stuffing-box at the top, clacks, windbore, and pumps, or rising main as it is sometimes termed (see figs. 204, 205).

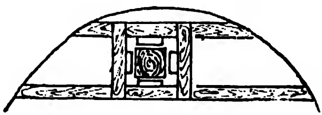


Fig. 208.—Collaring for Spears

The *spears*, *a*, are pitch-pine, arranged in equal lengths, and bolted together in a manner similar to those of the lifting sett, already described. They are connected to the engine at the surface, and to the plunger or ram at the bottom; and in order to give them a steady motion and prevent vibration, they are made to work through collarings and buntons (see fig. 208), fixed at intervals in the shaft, to which the pumps are also secured. Sometimes cast-iron guides are placed for the spears to work through.

The *plunger*, *b*, is usually made of cast iron, and is cast hollow. It should be accurately turned, and ground dead smooth so as to work as easily as possible and without unnecessary friction in the stuffing-box. When the water is of a corrosive nature, the plunger may be

encased in brass. The plunger works through a stuffing-box at the top, which must be air-tight and water-tight. When it works in the vertical line, there is very little wear or derangement, and to keep it in working order it is only necessary to renew the packing occasionally.

Clacks, *c*.—There are two clacks, and each has a separate chamber or clack-piece, with a bolted door by which admission is gained to them. One clack is placed in the suction leg, and the other in the delivery. They are similar in construction to the clacks of the lifting sett, and are made of brass or cast iron, with wrought-iron falls grathed or faced with leather. Cornish double-beat valves are also used.

The *windbore*, *d*, is flat-bottomed, and may rest upon the pit bottom or upon strong timbers or girders.

The *pipes* or *rising main*, *e*, do not differ from those of the lifting sett.

Pit-work Arrangements.—The forcing sett sometimes pumps from the shaft bottom, and sometimes from an intermediate point. In the latter case the water is first pumped up from the bottom by a lifting sett to the point where the forcing sett is to work. When the forcing sett is applied at the shaft bottom, the windbore is set upon the bottom; the clack-pieces are bolted on to the top of it, and above them, the pipes, forming the rising main, follow. When it is applied at some intermediate point, strong timbers or iron girders are placed across the shaft to support the whole weight of the pumps. The working barrel is connected at the bottom to the clack pieces by a short branch pipe, as shown in figs. 204, 205.

The spears are passed through collarings placed at intervals of 18 feet up the shaft; the rising main also rests upon collarings. Buntons are let into the shaft sides, and cross pieces of timber are arranged to form the collarings.

To prevent the spears descending too far in case of a misstroke of the engine, and also to prevent them falling

down the shaft in case of breakage, side or "banging" pieces of timber are bolted or strapped on to the spears at intervals down the shaft. When the engine makes its regular downstroke, these side pieces just touch the top of cross balks or girders which are fixed into the sides of the shaft; but in the event of a misstroke they rest upon the balks and prevent further descent and injury to the pumps and engine.

When the weight of the spears and plunger exceeds that of the water, balance bobs are required at the end of the engine beam or quadrant, in order to equalize the weights.

Action of the Forcing Sett.—The action of all force-pumps is very simple. The first process, that of raising the water from the sump or cistern, depends upon atmospheric pressure; and the second process, that of raising the water from the working barrel up through the rising main to the surface, depends upon the "force" of the downward stroke of the spears and plunger. As the plunger rises in the working barrel with the upstroke of the engine, a more or less complete vacuum is formed, and the pressure of the atmosphere upon the surface of the water in the sump causes water to rise up through the windbore and bottom clack into the working barrel, which its fills. When the plunger commences its downstroke, the bottom clack closes, and the water is *forced* up through the discharge valve into the rising main to the surface. Both of the clacks or valves open to allow the water to pass up, but close to prevent any water falling down. Thus with every upstroke of the plunger the working barrel is filled with water from the sump or cistern; and with every downstroke the working barrel is emptied of its water, which is forced into the rising main. When the plunger works at some intermediate point in the shaft, it pumps from a cistern, into which the lifting sett below has raised the water.

Underground Forcing Pumps.—The engine to work a lifting or a forcing sett is situated on the surface,

but with the underground forcing pump the engine is placed at the bottom of the mine, and consequently spears are entirely dispensed with.

The underground engine and pump must necessarily *force* the water to the surface. These generally consist of a steam cylinder, and a pump cylinder or working barrel, fitted with a through piston-rod, to which is attached a piston in the steam cylinder, and a plunger or ram in the working barrel. They thus work direct, and are usually double-acting, that is, pump water both with the forward and the backward stroke. They are made either vertical or horizontal, and are sometimes fitted with a fly-wheel.

The pump is generally placed at a convenient point near the shaft, so as to be as close to the water in the sump as possible. Sufficient height and width of engine-room must be provided by removing top or bottom stone, and in most cases the engine-room is arched with bricks.

The motive power employed for working the engine may be steam, compressed air, or electricity.

If steam is used, it may be conveyed in pipes down the shaft from the surface boilers; or boilers may be fixed underground to supply the engine. In taking steam from the surface there is usually much loss from condensation, however well the steam-pipes may be protected; yet this method is preferable to having boilers underground. The fixing of boilers underground is very costly; they are generally built in the mine, and a large space must be provided for them, as a rule, near to the upcast shaft. While there are still some mines at work with the foregoing arrangement, the prohibition of furnace ventilation by the 1911 Mines Act will prevent this method of providing power being now applied.

Turbine Pumps.—Owing to low first cost, ease of handling, and high efficiency, the turbine pump is now rapidly taking the place of the old forcing sett for mine

pumping. The pump consists of a series of revolving wheels called impellers, set side by side on the same shaft. The water passes into the centre of the impeller and is thrown off at the periphery, where it passes through a series of gradually enlarging openings into an outer casing. The effect of the decrease of velocity is to increase the pressure energy in the water, which now passes into a second chamber to be again driven forward by another impeller, and so on, until the pressure energy is sufficient to overcome the head through

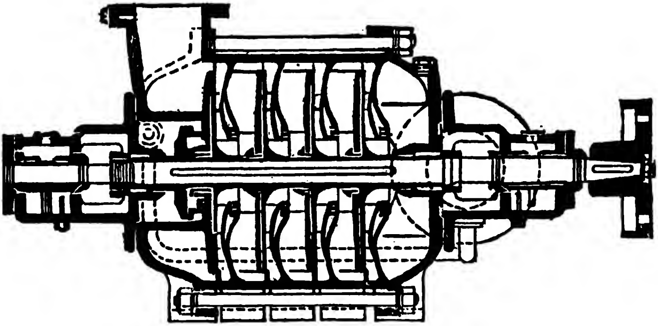


Fig. 209.—Section of Four-stage Turbine Pump

which the water has to be raised. A section of a pump of this class is shown in fig. 209. While admirably suited for dealing with large volumes of water, they are not easily constructed to deal with small quantities, especially against high heads, nor are they suitable for pumping from advancing workings such as a dook face, owing to difficulties of working them when they begin to draw air along with their water.

Compressed air is much used as a motive power for underground engines, especially in deep and fiery mines. The air compressor is usually placed on the surface, and the compressed air carried down the shaft in pipes. Compressed air is absolutely safe for transmission and

use in mines. It produces no ill effects upon the shafts or roadways; in fact, the exhaust air may be used for ventilative purposes. It is, however, usually a very expensive motive power. The first cost of machinery to compress the air, and the working cost, are high; then the available power or useful effect is very low, often not more than 25 per cent. Consequently compressed air is unfavourably looked upon as a motive power.

Electricity is now being successfully applied for pumping mine water. The dynamo for generating the electricity is placed upon the surface, and the current is conducted to the motor at the pump by a copper cable, which can be carried in any direction. (See Chap. XXXIII.)

CAPACITY OF PUMPS.—To calculate the quantity of water a pump is capable of delivering, it is necessary in the first place to ascertain by measurement the diameter of the bucket or plunger and the length of the stroke. The quantity of water delivered at each stroke should be equal to the area of the bucket or plunger multiplied by the length of the stroke. This gives the quantity in cubic inches or feet, which must be multiplied by 0·0036 or 6·23, the gallons in a cubic inch and cubic foot, respectively, to ascertain the number of gallons.

EXAMPLE.—Find the quantity of water in gallons which a pump 12 inches diameter, with a 6-foot stroke, is capable of delivering.

Diameter^2 × 0·7854 = area of pump.
Area of pump × length of stroke = quantity.
$12^2 \times 0{\cdot}7854 = 113{\cdot}0976$ = area of pump.
$113{\cdot}0976 \times 6 \times 12 = 8143$ cubic inches.
$8143 \times 0{\cdot}0036 = 29{\cdot}3$ gallons of water per stroke.

When the quantity of water delivered by each stroke of the bucket or plunger is ascertained, the quantity delivered per minute may obviously be found by multiplying by the number of strokes in a minute. Thus, in the above example, if the pump worked 6 strokes per

minute, 29·3 multiplied by 6 gives 175·9 gallons delivered per minute.

By means of the following formula, the quantity delivered by a pump may be ascertained more readily than by calculating in the manner above.

If D = Diameter of pump in inches,
L = Length of stroke in feet,
N = Number of strokes per minute,
G = Gallons delivered per minute,

$$G = 6{\cdot}23 \times \frac{0{\cdot}7854\ D^2}{144} \times LN = 0{\cdot}034\ D^2LN.$$

Thus, in the example previously given—

$$G = 0{\cdot}034 \times 12^2 \times 6 \times 6 = 176 \text{ gallons per minute.}$$

When the quantity of water to be raised per minute from the mine is known, the size of pump required can be ascertained by the converse of the above formula.

$$D = \sqrt{\frac{G}{0{\cdot}034\ LN}}.$$

Using the same example—

$$D = \sqrt{\frac{176}{0{\cdot}034 \times 6 \times 6}} = \sqrt{144} = 12 \text{ inches.}$$

The quantity of water a pump is capable of delivering, as determined by calculation, is the theoretical quantity. A pump does not work in such a perfect manner as to give the actual quantity due to its diameter, length, and number of strokes. There is always a certain amount of leakage at the suction and delivery valves as they are in the act of closing, and very often when closed, owing to imperfection of the valves. This leakage, or *slip* as it is often termed, is a variable quantity, depending entirely on the condition of the valves. If they are in efficient working order, the quantity slipping should not amount to more than 5 per cent; but if they are worn, or prevented by any obstruction from working properly, the amount may be a large and serious one.

To Ascertain the Pressure of Water Columns.—At mines

it is very often necessary to determine the pressure per square inch of water in pipes, pumps, old workings, bore-holes, &c., due to a head or column of water. To find this the depth in feet of column of water must first be ascertained. One cubic foot of water = 1728 cubic inches, weighs 62·4 lb., and a column of 1 sq. inch area and 1728 inches or 144 feet in height weighs or exerts a pressure of 62·4 lb. From this the following rule is got:

$$\frac{\text{Head of water in feet} \times 62{\cdot}4}{144} = \text{pressure per square inch,}$$

$$\text{or} \quad \text{Head of water} \times 0{\cdot}434 = \text{pressure per square inch.}$$

Thus, supposing, in the former example, the water had to be pumped to the top of a shaft 300 feet in depth, the pressure per square inch upon the plunger would be:

$$\frac{300 \times 62{\cdot}4}{144} = 130 \text{ lb.,}$$

$$\text{or} \quad 300 \times 0{\cdot}434 = 130 \text{ lb.}$$

If it be necessary to know the total pressure upon the plunger, in order to calculate size of engine required to do the work, it may be found by multiplying the area of the plunger by the pressure per square inch.

The area of the 12-inch pump is 113·0976.

$$113{\cdot}0976 \times 130 = 14702{\cdot}6 \text{ lb. total pressure.}$$

The size of engine cylinder necessary to overcome this pressure and pump the water may be easily ascertained, assuming an effective pressure of steam of 50 lb. per square inch:

$$\frac{14702{\cdot}6}{50} = 294 \text{ sq. inches of cylinder.}$$

One-half of this area should be added as an allowance for friction.

$$\tfrac{1}{2} \text{ of } 294 = 147, \text{ this added to } 294 = 441,$$

$$\text{then } \sqrt{\frac{441}{0{\cdot}7854}} = 23{\cdot}7, \text{ say 24 inches as the required diameter of steam cylinder to do the work.}$$

Horse-power to drive Pumps.—The number of foot-pounds of work done by a pump is obtained by multiplying the number of pounds of water raised by the height in feet through which it is lifted.

Since a gallon of water weighs 10 lb., then

$$\text{H.P.} = \frac{G \times \text{height of lift in feet}}{33{,}000}.$$

EXAMPLE.—Find the horse-power required to drive a pump of 10 inches diameter and 2 feet stroke when making 20 effective strokes per minute, and delivering its water against a head of 300 feet. Take the efficiency as 60 per cent.

$$\begin{aligned} G &= 0{\cdot}034 \times D^2 \times L \times N \\ &= 0{\cdot}034 \times 10 \times 10 \times 2 \times 20, \\ \text{and H.P. in water} &= \frac{0{\cdot}034 \times 10 \times 10 \times 2 \times 20 \times 300}{33{,}000}. \end{aligned}$$

But as only 60 per cent of the horse-power is available to raise water, the

$$\begin{aligned} \text{H.P. to drive pump} &= \frac{0{\cdot}034 \times 10 \times 10 \times 2 \times 20 \times 300 \times 100}{33{,}000 \times 60} \\ &= 123{\cdot}6. \end{aligned}$$

CHAPTER XXXII

THE DRAINAGE OF MINES—CONTINUED

Removal of Water from Dip Workings

Water in Dip Workings.—In many mines, part if not all of the coal to be worked lies at a lower level than that of the shaft bottom; this may be owing to the natural inclination of the strata, to faults, or to swellies. Any water given off in the mine, as the coal is being worked at these low-lying points, naturally flows to the lowest of the workings, where it accumulates, unless some means be employed for its regular removal. Usually steps are taken to remove the water to the shaft bottom, from whence it is raised to the surface;

but occasionally it is pumped direct from the workings to the surface.

The following are the methods employed to remove water from dip workings: Siphon, water tub, common lift pump, and force pump. The latter may be driven by men, horses, haulage ropes, steam, compressed air, water, petroleum, or electricity.

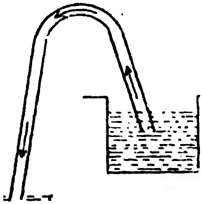


Fig. 210.—Common form of Siphon

Siphon.

Construction. — The siphon consists of a bent tube, or a number of pipes laid in the form of a bent tube. One of the legs of the tube is called the *suction*, and is placed in the water which is required to be removed; the other leg is called the *delivery*, and its end must be at a lower level than that of the water at the suction end. The siphon, therefore, does not raise water from a lower to a higher point, but takes it over a hill and delivers it at a lower point than that from which it started (see figs. 210, 211). At or near the suction end of the pipes is fixed a common clack to open upwards as the water flows up, but to close to prevent it flowing back at any time. At the delivery end a tap is fixed in the pipes, which is shut when the siphon is being filled with water previous to starting it into action, but is open when the

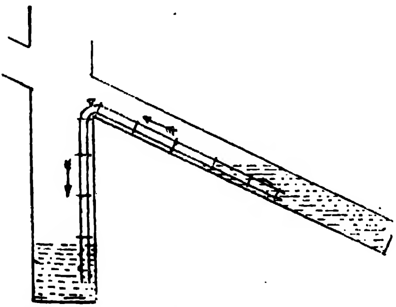


Fig. 211.—Application of Siphon to remove Water from Dip Workings to Shaft Sump

siphon is running. Before it can be started it is necessary that the pipes between the suction clack and the delivery tap be filled completely with water, or all the air pumped out. Every joint must therefore be perfectly air-tight, and for efficient working of the siphon all the pipes should be laid as straight and regular as possible.

Action.—The siphon, like the pump, is dependent for its action upon atmospheric pressure. Theoretically, it should raise water over a point 34 feet in vertical height, but in practice, owing to imperfect joints admitting certain quantities of air into the pipes, and the friction of the pipes, the vacuum is incomplete, and, as a result, 25 feet in vertical height is considered a very good suction. In all cases the delivery leg should extend to a lower point than the suction. After the siphon has been filled with water the delivery tap is opened, and the water in the leg flows out. A vacuum is thereby formed in the pipes, and the atmospheric pressure at the suction end causes the water to rush up the pipes to fill up the vacuum. The siphon, once started, will continue to work until all the water is removed, unless sufficient air gets in to stop the action.

Water Tub.—When the feeder of water is very small it may be removed by means of a tub which is filled with a bucket by a boy or man. When filled, the tub is moved along the rails to where the water will flow away, and there emptied by withdrawing a plug or opening a valve in the bottom.

Lift and Force Pump.

Men.—The common lift pump worked by hand is very often adopted to remove small quantities of water from dip workings, and by adding pipes on to the delivery end of the pump it may be made to force as well as lift water.

Horse.—When the quantity of water becomes too large, or the distance too far for the pump to be worked by hand, horses are sometimes applied. The horse is

attached to a projecting wood arm, and travels around in a circle. The arm is connected to a toothed wheel which works into a pinion wheel attached to a crank shaft to which the pumprod is connected. As the horse travels around the circle the wheel is revolved, and through the toothed wheels a reciprocating motion is imparted to the ram of the pump.

Hauling Ropes.—These are sometimes made to work over a clip pulley, which is keyed upon a shaft to which the pumprod is connected. When the ropes are in motion the pulley revolves, and this works the pump.

Steam.—This is sometimes conveyed inbye from the shaft to work an engine for pumping water, but owing to the disastrous effect of the heat upon the roof and sides of the air-ways, and to the loss from condensation, it is much objected to.

Compressed Air.—This is used in many instances. It is an excellent motive power, and may with perfect safety be transmitted to any part of the mine, but it is very costly owing to the high cost of the compressing plant and its low efficiency.

Water.—Where a sufficient head of water can be conveniently applied, a hydraulic pump may be used. The water-power to work the hydraulic pump may be obtained either from the column of pumps in the shaft, or conveyed from the surface, or from an upper seam. When the water has done its work it is exhausted or discharged into the delivery pipes, and is returned along with the water pumped by the hydraulic pump from the workings into the sump, from which it is raised to the surface by the pumping sett. Hydraulic pumps can be economically applied in many instances to pump water from dip workings to the sump, where steam would be quite inadmissible, and compressed air too expensive.

Petroleum.—Oil-engines have in recent years been employed for pumping small quantities of water from distant dip workings. The engines are similar in general outward form to gas-engines. The driving power is

obtained by the explosion at regular intervals of heated petroleum vapour in the engine cylinder. The explosion in some engines is produced by an electric spark, in others by the heat of the cylinder together with the compression of the vapour. The engine is geared to its work as required, very often by a belt. These engines, however, can be used only in situations in mines absolutely free from the presence of firedamp, because a large open lamp is required for the heating of the vaporizer in order to start the engine. The air passing over the engine and the exhaust gases should go straight into the returns. Under the Coal Mines Act, 1911, no internal-combustion engine shall be newly introduced into any coal-mine, the grave objection to this form of power being the formation of small quantities of carbon monoxide during combustion. This deadly gas escapes with the exhaust and enters the ventilating current, where it may give rise to an accident.

CHAPTER XXXIII

ELECTRICITY APPLIED TO MINING OPERATIONS

Electrical Terms.—The practical units used by electrical engineers to denote the pressure and flow of electric current through a conductor correspond to those used by mechanical engineers to denote the pressure and quantity of steam or water flowing through a pipe.

The terms are: the Volt, the Ampere, the Ohm, and the Watt.

The Volt.—This is the unit of electromotive force or pressure of an electric current. It is equivalent to the difference of potential (P.D.) between two points, and is usually denoted by the letters E.M.F.

In speaking of volts, the electrical engineer uses the term in the same way as the mechanical engineer uses pounds per square inch when referring to pressure of steam. When a dynamo is said to generate electricity at so many volts, or to give an E.M.F., the meaning conveyed is that electrical pressure is produced corresponding to the meaning conveyed by so many pounds per square inch pressure of steam at boilers. In the electrical rules the following voltage limits have been adopted for mines:

Limits of Pressure.	*Supply System.*
Not exceeding 250 volts.	Low pressure.
Between 250 and 650 volts.	Medium pressure.
Between 650 and 3000 volts.	High pressure.
Exceeding 3000 volts.	Extra high pressure.

The Ampere.—This is the unit of current passing through any conducting substance. To explain it by comparison we should say it is the measurement of electric current in much the same way as gallons may represent the flow of water through pipes. Whereas volts represent *pressure*, amperes represent *rate* of current flow of electricity.

The Ohm.—This is the unit of *resistance* to the flow of a current of electricity through any conductor or material through which it may be desired to pass it. The standard ohm is the resistance of a column of mercury at 0° C. of 1 square millimetre cross-section and 106·3 centimetres in length.

The legal volt is the electromotive force or *pressure* which maintains a *current* of 1 ampere in a conductor whose *resistance* is a legal ohm.

The Watt.—This is the unit of *power*, and is equal to 1 volt-ampere. It provides a practical way of applying the foregoing units, and it represents the work done by an electric current. The work done by a direct current is found by multiplying the number of volts by the

number of amperes. Thus a current of 100 amperes at an E.M.F. of 250 volts would give:

$$100 \times 250 = 25{,}000 \text{ watts.}$$

A kilowatt, generally written kw., is 1000 watts. Thus the machine which generated the above 25,000 watts would be described as a 25-kw. generator.

Electrical Horse-power.—746 watts equal 1 horse-power. Approximately, therefore, 1000 watts, or 1 kw., equals $1\frac{1}{3}$ horse-power.

To find the horse-power of a dynamo generating, say, 100 amperes at an E.M.F. of 250 volts:

$$\frac{100 \times 250}{746} = 33{\cdot}7 \text{ h.p.}$$

The engine required to drive the dynamo would have to be about 50 h.p., the difference being due to friction and other losses, both in the engine and dynamo.

The Electrical or Commercial Efficiency is the ratio of the power put into a generator from an engine to the electrical energy developed. The commercial efficiency of good machines should be 90 per cent.

$$\text{Commercial efficiency} = \frac{\text{useful work given out}}{\text{work put in}}.$$

Electric Currents.—These may perhaps be best studied by the aid of an analogy—that of a current of water. Water may be made to flow through pipes by means of a pump. The pump has to overcome the friction of the pipes, and the weight of the water due to the *head*. In the same way electric current may be said to be forced along a wire or conductor by the generator or dynamo. The dynamo supplies the force or pressure to cause a flow of electric current, and it has to overcome *resistance* in the conductor. This pressure is the E.M.F. or potential difference. Thus, if two points along a wire are at *different* potentials, a current will flow from the point at the higher potential to the one

at the lower, the higher being the *positive* electric current, and the lower the *negative*. It is convenient to have a starting-point or zero for our measurements of potential, in the same way as the zero for water is often taken to be the sea-level. We take the earth's potential to be 0, potentials above it being positive, and those below it negative.

The effects which electric currents may produce are:

1. The magnetic effect. A magnetic field is set up along the axis of a coil carrying current. An electro-magnet consists of a coil of wire carrying current, wound round an iron or steel core. Here it is necessary to explain the meaning and use of the terms north pole and south pole. If a magnet is freely suspended it will come to rest in a definite direction pointing north and south. On account of this the magnet's poles are called north and south respectively. The north pole of the electro-magnet is the end or pole at which the lines of force leave the core, and the end at which the line enters is called the south pole. The path of the lines is called the magnetic circuit.

2. The heating effect. The temperature of a conductor carrying electric current is raised. The ordinary incandescent electric light is an illustration of this effect.

3. The chemical effect. By the passage of electric current chemical compounds in solution are split up into their constituents.

Conductors.—The best electrical conductors are metals and their alloys. The poorest conductors are glass, porcelain, slate, marble, ebonite, mica, and rubber, and, being low in conductivity, they are used as insulators. Any material which readily allows the flow of electric current is said to have high conductivity and low resistance, and vice versa.

Alternating Current.—An alternating current is one which varies periodically and flows first in one

direction and then another. It varies also in magnitude. A complete alternation or *cycle* occurs during the time an armature conductor is passing a pair of poles, that is, a north and south magnetic pole. A number of such cycles takes place in a second, depending upon the speed and number of poles of the machine, and this number is termed the " periodicity " or " frequency " of the circuit. The frequency recommended by the Engineering Standards Committee is 50. If in any particular case a lower frequency is preferable, they recommend that 25 be adopted.

The alternating current may be either single-phase, two-phase, or three-phase. The first two are seldom adopted in mining.

By " three-phase " is meant that three separate currents are generated. The alternator has three distinct armature windings, so placed that the phase difference of current of each with respect to the others is one-third of a period, or otherwise the angle between each is 120°.

Direct Current.—A direct or continuous electric current is constant in direction and magnitude. The currents produced in the armature are induced by exactly the same actions as in alternators, and are themselves alternating. But they are " rectified ", or converted into *continuous* current, by a device known as commutating. The commutator is made of cylindrical shape, and consists of a number of hard-drawn copper segments insulated from each other by mica sheets. The ends of each armature coil are connected to segments, so that when the direction of the current in the coil is being reversed, the end of the coil in connection with a given brush is being reversed. In this way a steady current, whose fluctuations are quite imperceptible, is obtained in the external circuit.

Comparison of Alternating and Direct Current for Mines.—Alternators can be constructed for

high pressures, and alternating current is advantageous where electrical energy has to be transmitted over long distances, and may be used *direct* or *transformed*, as may be desired, thus reducing installation costs. Owing to absence of commutators in squirrel cage rotors, sparking is obviated, but if wound rotors with slip rings are used, sparking will take place at starting, just as readily as with D.C. On the other hand, motors start badly under load, and the choice of motor speeds is limited.

Direct current has the advantages of simplicity, with steady pressure under variable load. It is suitable for lighting purposes, and for motors which have varying load or speed, or have to start up under heavy load.

On the other hand, with direct current, motors may spark at the commutators; this, however, is dealt with by enclosing them in flame-proof casings when they are to be used in gassy mines. The voltage at which the current may be generated is limited.

ELECTRICAL MACHINERY

Dynamos or Generators.—These are the machines for generating electric current. The design is based on the fact, discovered by Faraday in 1831, that a current of electricity is generated in conductors by moving them in a magnetic field. This movement is produced by mechanical energy, and that energy is converted into electrical energy of a definite current at a definite voltage, depending upon the conditions of construction, speed of revolving, and the resistance of the circuit. As already stated, dynamos may generate either alternating current or direct current.

Alternating-current Generators or Alternators.—Between the poles of the " field magnets " revolve the coils, known as the armature, in which the currents are induced. The terms " stator " and " rotor " are usually used instead of field coils and armature. There are three main types of alternator:

1. The fixed magnet—rotating armature.
2. The fixed armature—rotating magnet.
3. The inductor type—both field and armature coils are fixed, and moving masses of iron set up the induced currents.

The rotating-magnet type is generally adopted, as it possesses many advantages.

A small, separate, direct-current dynamo—the exciter—is used to excite the field coils of the alternator. This auxiliary machine is mounted on the alternator shaft and driven by the same engine.

Direct-current Generators.—These are invariably of the revolving-armature type, and usually multipolar. The essential parts consist of:

(*a*) The field magnets.
(*b*) The armature.
(*c*) The commutator.

The armature, which, with its commutator, is mounted upon the spindle, and runs in the bearings of the dynamo, is free to revolve between the north and south poles of the electromagnets. Upon the armature being revolved, the field magnets are excited by current from the dynamo itself, and thus the E.M.F. is developed.

The current is collected from the commutator by carbon brushes and conducted by "mains" to the switch-board.

The Switchboard.—For low or medium pressure, boards are made of slate or marble slabs carried by steel framing, and they are fitted with all the necessary controlling appliances, measuring instruments, and safety devices. Switchboards may be either main or distributing, their chief function being to control the whole of the electrical energy developed by the generators. The main switchboard is fixed in the generating station, but distribution switchboards are frequently placed at the shaft bottom, or at some position inbye, where current is taken off to a group of motors.

The Electrical Rules contain complete instructions for erection and fitting up of switchboards, and should be carefully followed.

For direct-current switchboards it is necessary, in order to have proper control of the generator and circuits, to have the following fittings and instruments:

(*a*) A double-pole switch.

(*b*) A double-pole fuse.

(*c*) A single-pole switch with an automatic circuit-breaker which opens when the current exceeds the normal.

(*d*) A voltmeter to record the voltage or pressure.

(*e*) An amperemeter to record the current which the generator is producing.

(*f*) A field regulator, which is a resistance coil for regulating the voltage of the generator.

In the case of high-tension switchboards, special precautionary arrangements for safety have to be taken. Oil-immersed circuit-breakers are used, and all live parts are placed behind the switchboard, so that it is impossible for anyone to come in contact with them. In all cases of underground switchgear it is best enclosed in a flame-proof case; the various switches may be connected to a panel carrying common bus-bars, and the doors should be so arranged that it is impossible to switch on with the door open.

Cables.—The conductors, cables, or mains, by which the current is conveyed from the source of supply to the lamps or motors which require the current, are made of copper wire. The cable must be of sufficiently large sectional area to carry the current without undue heating. The larger the current to conduct, the greater must be the size or sectional area of the cable.

For long-distance, high-tension current transmission, bare cables, supported at intervals by glass or porcelain insulators fixed on poles, are sometimes used, and also for carrying medium-pressure currents about collieries

for surface work. Where, however, there is any possibility of workmen coming in contact with them they should be insulated.

In buildings cables are generally run in steel tubes to reduce risk of fire.

Cables are insulated by hygroscopic or porous materials such as jute or paper, or by non-hygroscopic or waterproof materials, such as rubber and vulcanized bitumen.

The jute- or paper-insulated cables are covered by lead, rubber, or vulcanized bitumen.

Cables are sometimes further protected by "armouring". This consists of either a double layer of steel tape or a layer of steel wires.

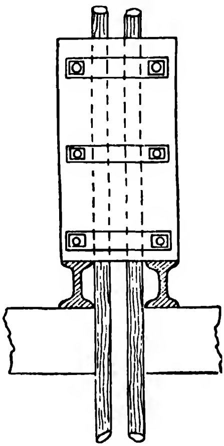


Fig. 212.—Methods of securing Cables in Shafts

The fixing of cables in a mine and running them along the roadways of mines requires careful and serious supervision, as the conditions are severe. For shafts and mine roadways the cables should be of the best quality and workmanship obtainable, and to comply with the Mines Act they must be armoured. In the shaft the cables, which must also be armoured, are suspended:

(*a*) By fixing them in strong wood boxing from the top to the bottom of the shaft. The boxing should fit tightly around the cable, and be made practically watertight by coatings of pitch and tar.

(*b*) By means of cleats resting on cross beams, and placed at such intervals as are necessary to support the weight of the whole length of cable in the shaft. (See fig. 212.)

Lead-covered cables are unsuitable and unsafe in wet

shafts, as water has an injurious action upon the lead.

The risks to be guarded against, both in shafts and along the roadways, are:

(*a*) Liability of electric shock to workmen.
(*b*) Risk of setting fire to combustible material.
(*c*) Possibility of igniting gas.

Cables in the mine must be efficiently protected from mechanical damage, and supported at sufficiently frequent intervals and in such a manner as adequately to prevent

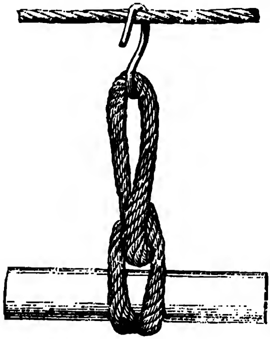


Fig. 213

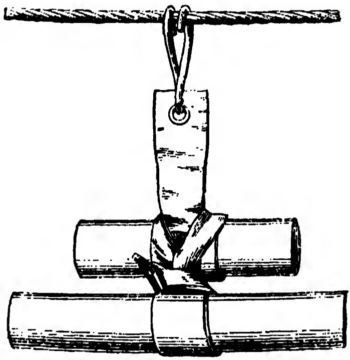


Fig. 214

danger and damage to the cables. Concentric cables or two-core cables, protected by a metallic covering, must be used where the roadway is also used for mechanical haulage, and where there may be risk of igniting gas or coal-dust.

Cables along the roadways in the mine are suspended from roofs, sides, or timber in such manner as to yield when struck by a fall of stone. The manner of suspension and materials used should give sufficient strength under ordinary circumstances, but should yield before serious damage occurs in the event of a fall of stone (see figs. 213 and 214).

Subsidiary cables, in the confined and circuitous passages of the inner roadways of the mine, are subject to more wear and tear and damage from falls than main-road cables, and therefore require more care and attention.

Trailing cables attached to coal-cutters at the working face are liable to many accidents. They have of necessity to be flexible, and at the same time heavy enough to carry the large current required. The insulation should be strong enough to allow of the cable being trailed along the rough floor of the mine. Sometimes the trailing cable is passed through a hosepipe, and in other cases an outside covering of leather or rope alone is relied upon for protection, but the principal class of trailing cable is covered with a sheath of tough rubber and is generally known as cab-tyre trailing cable. All trailing cables must carry a third core if for D.C., and a fourth if for three-phase A.C., to form an earth connection between the machine and the armouring of the main cable. The trailing cable should always be wound upon a drum when not in use.

Motors—Direct-current.—One of the chief uses of electric current is to drive machinery in place of employing steam-engines direct. The mechanical energy of the steam-engine is converted into electrical energy by the dynamo, and this electrical energy is conveyed through conductors, and, by means of motors, is reconverted into mechanical energy. The electric current produces rotation of the motor armature, which, connected by direct coupling, or by belting, actuates the machinery. For starting the motor a special switch is used which first inserts resistance in series, and then gradually cuts it out as the motor speeds up. The armature revolving in a magnetic field induces an E.M.F. in the armature coils which is opposite in direction to the supply pressure. This back E.M.F. is proportional to the speed, and the current taken by the motor will be equal to the net voltage divided by the resistance of the

armature. Therefore, in order to avoid burning out the armature coils, the starting switch is adapted to put on the current gradually as the motor speeds up.

Motors may be series-wound, shunt-wound, or compound-wound, the winding depending upon the nature of the work the motors are required for.

In a series-wound motor the field coils consist of comparatively few turns in series with the armature (see fig. 215), and the excitation thereby varies with the load. It possesses great starting power, and runs quickly when

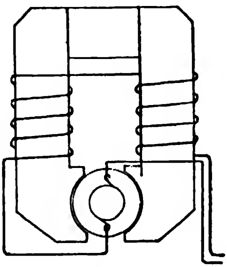


Fig. 215.—Series-wound Machine

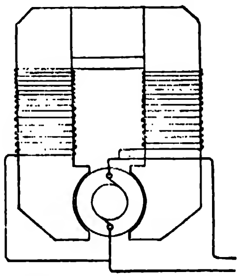


Fig. 216.—Shunt-wound Machine

lightly loaded, and slowly when heavily loaded. It is therefore specially suitable for driving machines which have to start under heavy load, as in pumping, hauling, or coal cutting, when geared to the machines by toothed wheels.

In the shunt motor the field coils are connected to the motor terminals, and are thus in parallel with the armature (see fig. 216). They consist of a large number of turns carrying a small and constant current. The shunt-wound motor has not such great starting power as the series-wound, but when supplied with current at a constant voltage will run uniformly in speed under a varying load, the power absorbed being proportional to the work done.

The compound-wound motor has series coils with shunt winding (see figs. 217, 218). The starting power

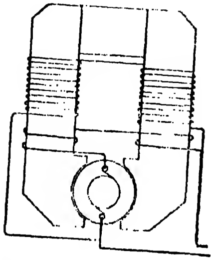


Fig. 217.—Compound-wound Machine with Long Shunt

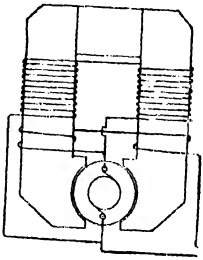


Fig. 218.—Compound-wound Machine with Short Shunt

of the shunt motor is thereby improved, although at the expense of the speed regulation. The compound-wound

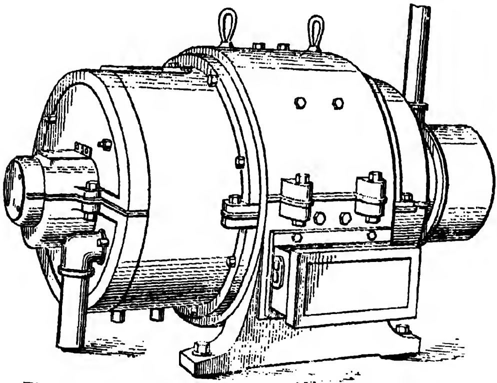


Fig. 219.—Totally-enclosed Motor with Ventilating Pipes

motor possesses, to a certain extent, the merits of the two former without their disadvantages.

Motors are made in four patterns for mining work—open, protected, ventilated, and totally enclosed—depending upon the position and nature of the work. When they have to work in dusty and damp places, totally-enclosed motors should be adopted, and in certain parts of mines these should be made explosion-proof. They should always be of strong mechanical design for mines, and made above the power required; otherwise there may be trouble from heating and sparking.

Alternating-current Motors.—There are two types of alternating-current motors in use:

(*a*) Synchronous motors,
(*b*) Non-synchronous motors,

and these may be either single-phase or polyphase. The former type is never used in mining.

Non-synchronous or induction motors (see fig. 220 on plate) consist of a stator, a ring-shaped iron shell in which are fixed the magnetizing coils. Inside the shell is the rotating armature or rotor, which, in its simplest form, consists of a series of transverse copper bars, joining corresponding points on the circumferences of two parallel copper rings, the whole being mounted on, but insulated from, a laminated iron core. This is the squirrel-cage type. Its resistance is naturally low, and at starting might be dangerous to a large machine. To avoid this another form of rotor, the slip-ring, is often employed, in which external resistance can be thrown into circuit by carbon brushes on the slip rings while the motor is speeding up, and then cut out again down to a short circuit.

Running of Motors.—With all motors, whether direct-current or alternating-current, it is required to have subsidiary apparatus near at hand for the proper control of the electric energy, and the running of the motors. The following are necessary:

1. To turn on or off the current: (*a*) by a *switch* or starter under the control of the person in charge; (*b*)

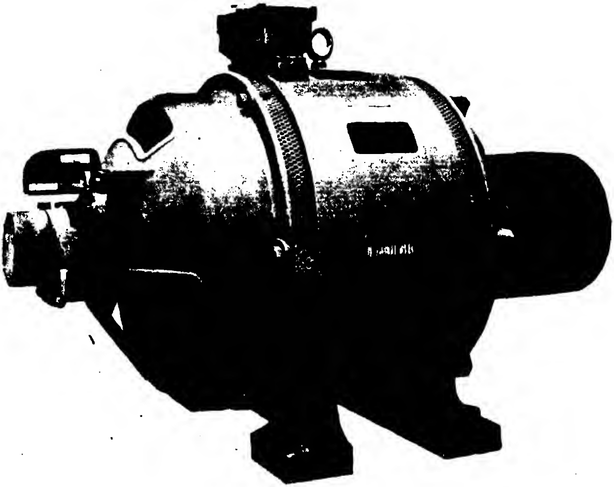


Fig. 220.—Three-phase Motor

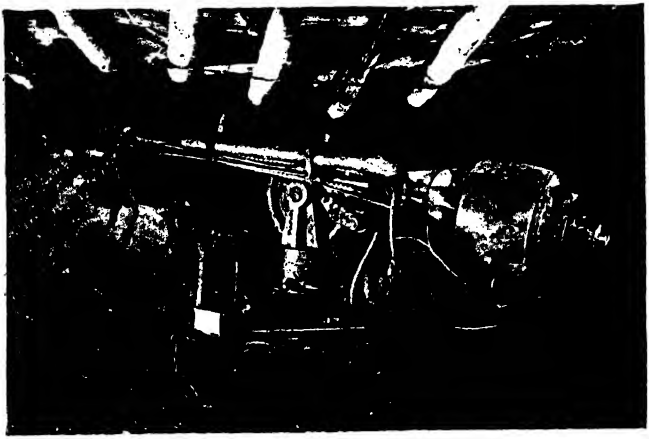


Fig. 221.—Motor-driven Rock Drill

by an *automatic circuit breaker* which operates a switch automatically when the current exceeds a definite amount; or (*c*) by *fuses* which automatically break the circuit when the current is too large.

2. Instruments for measuring current and voltage. Motors require frequent examination to keep them in good running order. The rules require that "The pressure shall be switched off apparatus forthwith if open sparking occurs, and during the whole time that examination or adjustment disclosing parts liable to open sparking is being made. The pressure shall not be switched on again until the defect has been remedied." When a motor develops sparking at the brushes it will probably be caused by one of the following: bad contact between brushes and commutator, which may be due to dirt or insufficient pressure, or to a worn uneven surface of commutator, overloading of motor, or an imperfect armature coil or connection. Totally enclosed motors which are used underground have to be specially designed for such work in order to avoid overheating. The rules require "All motors shall be constructed so that when any part is live all rubbing contacts are so arranged or enclosed as to prevent open sparking". This applies where gas may occur in quantity sufficient to be indicative of danger.

It is important that all motor frames and transformer cases underground should be effectively earthed. This also applies to all armoured cables, and the lead or copper sheathing of cables.

DEPARTMENTS IN MINES WHERE ELECTRICITY IS APPLIED

The departments of mining to which electricity is applied are various and gradually increasing. In some departments its use as a transmitter of energy is rapidly widening as improvements are introduced and better results obtained. The uses to which it is applied com-

prise lighting, signalling, drilling, shot-firing, coal-cutting, pumping, hauling, and winding. These are the chief operations which are conducted by the application of electricity; but there are other minor purposes to which it has been applied, and may be increasingly applied in the future. Many of the operations enumerated are carried on in the same mine simultaneously and independently of each other, although the nature of the operations differ widely, and the currents are generally supplied from a system of cables from one generating-station. It is as a transmitter of power or energy from the surface to distant points in the mine, for such purposes as coal-cutting, hauling, and pumping, that electricity is making headway, and likely to do so in the future.

Advantages and Disadvantages.—Compressed air is perhaps used more extensively as a means of transmitting energy from the surface to operations underground than any other, chiefly owing to its safety. It is, however, a costly method on account of its low efficiency. The establishment of the system, and also its maintenance and distribution, are expensive. It is also a difficult and inconvenient system in thin seams where large and heavy pipes are to be used. Electricity has the advantage in the ease with which it may be transmitted along flexible cables, which can be easily adapted to the circuitous roadways of a mine, and handled without much difficulty in thin seams. Its first cost is less, and a higher total efficiency is obtained in comparison to compressed air, especially in long distances underground. The cost for maintenance is also usually less.

On the other hand, electricity is not regarded with favour for use in those parts of a mine where safety-lamps are required, in consequence of the presence of firedamp. There is always present the possibility of sparking, and thereby exploding a gaseous mixture, or of damage to the insulation of the cables and thereby

causing a fire. These are serious risks, but the occurrences are avoidable by the use of good material, good workmanship, and having the cables and motors well protected and regularly examined. It must of course be kept in mind that if gas accumulates the workmen must be withdrawn until it is cleared, and an electric motor is not intended to work in an inflammable atmosphere any more than a flame safety lamp is. There is also the danger of men being injured by electric shocks from contact with wires and so receiving the current, but this danger is guarded against by having the machinery and wires or cables properly insulated.

Lighting.—Electricity is now largely used for illuminating the surface arrangements, such as engine-rooms, workshops, heapstead, offices, &c. The cables are also usually suspended down the shafts, and the current used for lighting the pit bottom, shaft sidings, underground engine-houses and stables, and also in some cases for lighting considerable distances along the main haulage roads. The lamps used are generally the incandescent type, from 16 to 50 candle-power, and are fixed at such places and at such distances apart as the circumstances of the mine require. The advantages of the electric light at pit bottoms are obvious. The light is absolutely safe, and produces neither smoke nor deleterious gases. It is in every way superior to gas-light, or paraffin lamps, which have no doubt on many occasions been the cause of surface and underground fires at collieries.

Wherever electric lighting is installed underground, only well-designed switches and fuses should be adopted. The wiring of the lighting circuits should have special attention. The wires may be arranged either (*a*) open-type, supported on porcelain insulators, or (*b*) in steel piping to contain the wires. In the former case they are open to inspection, and readily repaired or renewed when necessary; in the latter they are protected from damage from dampness or haulage accidents.

The sinking of shafts may now be facilitated very much by the use of the electric light, which may be suspended in the shaft to a short distance above where the sinkers are working. The light may be of such power as is required to sufficiently illuminate the bottom of the shaft. Arrangements are made to lower the light as the sinking progresses, and also to raise it to a point of safety when blasting operations are being conducted.

Signalling and Telephoning.—For communicating signals in winding operations in shafts, electric bells are now largely adopted, and are found of considerable advantage over the old-fashioned hammer signal or rapper, especially in deep shafts where rapid winding and large quantities of coals have to be dealt with.

The electric currents for this purpose are usually supplied by large Leclanché cells, and the conducting wires are run in one insulated cable and secured at intervals to the shaft sides; sometimes the cable is covered by a grooved wooden casing to protect it from injury; and occasionally the bare wires are suspended from the headgear in dry, roomy shafts.

Various kinds of electric signalling apparatus with automatic indicators, designed to avoid accidents from misunderstanding of signals, are in use on the Continent, and some type of automatic signalling is now compulsory in this country. One system consists of ordinary single-stroke bells supplemented by luminous indicators, so that the audible signals may be confirmed by visible indicators. The switches are so arranged that no signal, except the emergency stop signal, can be indicated in the engine-room unless the corresponding switches at the top and bottom of the shaft are set to give the same signal, so that in the event of the onsetter being ready first, and the signal being given by him, it cannot be indicated in the engine-room until the banksman has also given the corresponding signal.

Electric bells are also largely used on main roads where mechanical haulage is in operation. On long

roads, and where there are many branch roads, the electric bells are quicker, and less liable to get out of order, than the old-fashioned rapper arrangement. Either bare or insulated wires may be adopted, but on dry roads bare wires are preferred and generally used, because a signal may be sent from any part of the road by connecting the two wires with a metal rod, or otherwise bringing them into contact. The wires are usually run along one side of the road near the roof, and are stretched from insulators screwed on to wood plugs, which have been driven into holes bored into the roof at from 12 to 15 yards apart, or attached to timber. Where bare signal wires are used in safety-lamp mines the voltage should not exceed 25 volts. Where insulated wires are adopted on wet roads, signals or telephone messages can only be communicated from fixed signalling-stations properly equipped.

Where electric signalling bells are in use, telephones are generally fitted up, the same wires answering both purposes. Messages can in this way be communicated from stations on the surface, such as engine-rooms and heapsteads, to shaft bottoms and other parts of the mine. They are very useful on underground hauling roads, especially in large mines where the roads extend long distances and are widely apart. Fixed telephones are usually placed at important stations on the main underground roads for dispatching messages, but portable telephones are also carried by officials to attach to the bare wires on any part of the road, by means of which messages may be transmitted or received. Where the wires are insulated, this of course cannot be done.

Drilling.—For drilling shot-holes in coal or stone, electrically-driven drilling or boring machines have not been much adopted yet. They have been tried in tunnelling and shaft-sinking with some amount of success, but not in coal-seams for ordinary work. They appear, however, to have been more determinedly introduced in ironstone mines, and at the present moment

rock-drills driven by electricity are in regular use in some of the Cleveland ironstone mines for drilling the ordinary shot-holes in the hard ironstone (see fig. 221 on plate). An electric rock-drill was specially invented for this purpose by Mr. A. L. Stevenson, and appears to give satisfactory results.

A new type of electric coal-drill has been recently introduced for rapidly drilling shot-holes in a long-wall face which has been undercut by a coal-cutter. The total weight of the appliance is only 180 lb. The frame is made in different lengths to suit the height of various seams. The body of the drill is mounted on trunnion brackets with hinged lids, so that it may be readily taken to pieces for moving about the pit. The brackets slide up and down the sides of the frame, and the drill may be put in any vertical direction within a range of more than 90 degrees. The motor is series wound, and the connections are made by putting the cable sockets on to the drill plugs fitted with a bayonet arrangement. Three drills are supplied, and when drilling in coal a hole 2 inches diameter and 6 feet long can be finished in about 2½ minutes, including the changing of the drills.

Shot-firing.—The firing of shots, or, in other words, the igniting of charges or explosives by electricity, has been adopted more or less in mines for many years, but its adoption has become almost universal during the last three or four years. This has been brought about by the increased use of the higher explosives, necessitated by recent legislative enactments on the use of explosives in mines.

Two kinds of detonator fuses are employed, known as low-tension and high-tension. In these fuses sufficient electricity is employed to generate heat to cause the detonators to explode, and they in their turn explode the charges of explosive in the shot-hole. In the low-tension fuses the heat generated is due to the resistance of a small loop of high-resistance wire embedded

in the mixture contained in the detonator. In the high-tension a spark is produced which passes between the ends of two copper wires embedded in the detonator mixture.

The electric current for firing the low-tension fuses is produced by a small portable battery of dry cells or by a small magneto. For the high-tension fuses the current is generated by means of a small portable magneto-electric machine, and more rarely by utilizing the current from the dynamo mains where electric-lighting or other electric apparatus is used.

In the operation of firing a shot, the cables used are long enough to reach from a place of safety, where the operator stands, to the wires attached to the detonator in the explosive charge at the back of the shot-hole. The current is not turned on until all persons in the vicinity of the shot have reached a safe place. The explosion is instantaneous when the current is turned on.

Electric shot-firing has the advantage of being safe to all persons using it. No sparks are produced in the outside atmosphere, and as soon as the machine is disconnected there is no danger in returning to the shot-hole after an interval of ten minutes if the charge from any cause has not been exploded.

In sinking pits the shots are now generally fired by electricity, it being more expeditious and efficient, and certainly a great deal safer than the old system of train-fuse. The cables of copper wire are properly insulated, and are wound on drums on the surface, from which they are hung down the shaft, and can be lengthened by unwinding when required as the shaft deepens. When it is intended to fire the shots the cables are connected to the wires of the detonators, and one shot may be fired, or a number simultaneously, by an operator on the surface after all the sinkers have been withdrawn from the shaft.

Very often, when deep shafts are being sunk, an electric-lighting plant is erected for the purpose of light-

ing the surface arrangements and to supply the light in the shaft for the sinkers. Sometimes the current has been tapped and utilized for the purpose of firing a number of shots in the bottom simultaneously. This practice, however, was attended by considerable danger, and to ensure safety special precautions were necessary. Owing to this risk the Electricity Rules prohibit the use of current from lighting or power circuits for firing shots.

Coal-cutting and Coal-conveying.—There are some seams of coal which are so difficult and unprofitable to work by hand labour, by reason of the thinness or hardness of the coal, or both combined, that, to enable them to be worked to advantage, machines with appliances to undercut the coal have been introduced, and are now being used in various parts of the country, in some cases with considerable success. In this country the machines mostly in use are for longwall faces, along which they travel.

The power to drive the coal-cutting machines is usually either compressed air or electricity, the latter being now generally preferred. Compressed air is costly, and is difficult to transmit long distances, especially in thin seams, owing to the heavy pipes required for the air being difficult to handle, and often getting broken where there is much heaving of the floor. Electricity, when it can be applied at the face without any danger from firedamp, is certainly preferable, owing to the comparative ease with which the electric cables can be laid into the face of the workings along the devious roadways of a coal mine.

The noise of the working of the machinery of coal-cutters being an objectionable feature to their use, the adoption of electricity has a considerable advantage in this respect compared with compressed air; the exhaust of the latter makes a great deal of noise and dust, and renders the work of the attendants more difficult and dangerous.

The coal-cutting machines are supplied with the

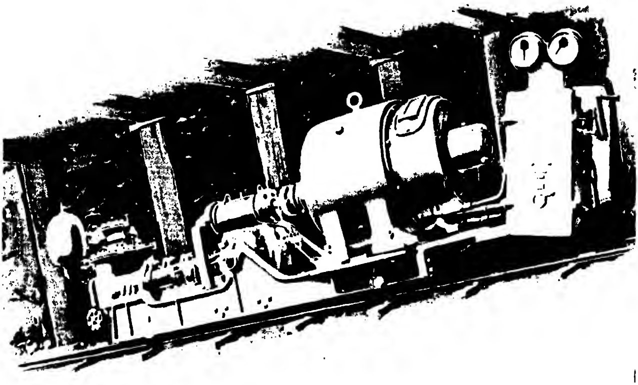


Fig. 222.—Pump with Direct-current Motor for Dip Working

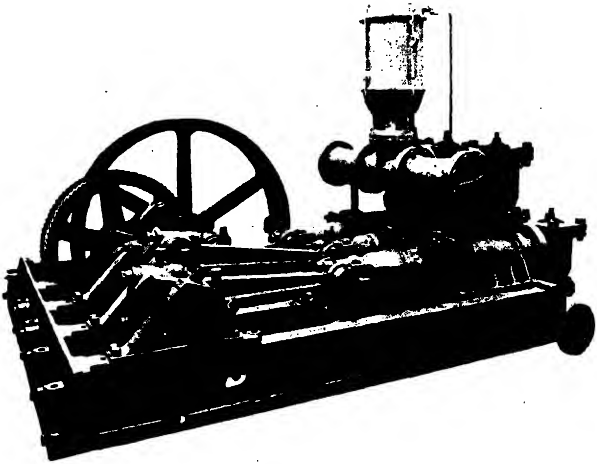


Fig. 223.—Motor-driven Three-throw Horizontal High-lift Pump

electric current either from a plant erected for the special purpose, or, more frequently, from the generating plant to supply the colliery generally. Usually the starting switch and resistance is embodied in the general design and carried on the machine, the current being provided by suitable trailing cables which are connected to gate-end boxes fitted to the permanent supply.

Coal-conveyors, described in Chapter XV, are often used in conjunction with coal-cutters, and are driven electrically, the motor being placed on the main gate road, and moved forward with the conveyor. As the motor has to work under unfavourable conditions, viz. amongst dust and near the coal face, where the air is warm, it requires to be rated much above its work and carefully attended to.

Pumping.—Electricity is now largely employed for driving pumps to deal with mine water.

In sinking pits, pumps with motors attached are hung in the shafts, and can be raised and lowered as required.

At shaft bottoms, electrically-driven ram pumps or high-speed turbine pumps are used to force the water to the surface. The high-speed turbines have lately come into use, and are being adopted in many cases. Having to run at high speeds, both motor and turbine require to be well made to withstand the stresses produced by centrifugal force.

As a means of transmitting energy to long distances underground, electricity has been more extensively put to the test of practical application for pumping water from inbye dip workings than for any other purpose.

Pumps fixed on cast-iron frames on wheels, designed to run on the underground tramways (see fig. 222 on plate), are used in mines in all coalfields for following down water which is being pumped out of dip workings, or to follow the advancement of the faces of the workings where water is given off. The frame not only carries the pump, but the electric fittings, consisting of the motor with starter, and usually two drums of cable.

The latter are for lengthening the cables to allow the pump to be moved forwards when the water recedes by being pumped out.

Fixed electric pumps are also largely used for pumping water from standages in workings to the shaft. Many of these stationary pumps are of large size, and deal with heavy feeders of water. They may be either of the ram (see fig. 223 on plate) or centrifugal (see fig. 224 on plate) type; but, whichever is adopted, both pump and motor should be of ample capacity in order to avoid continuous running and prevent over-heating of the motor.

It is desirable that the electric pump room should be ventilated with intake air.

Electric Haulage. — The haulage of coal underground is now largely carried out by electricity.

Auxiliary haulage.—The adoption of electric motor haulage and distant inbye hauling to the main stations is now much adopted. It is especially useful for hauling out of dip places, particularly where only the main-rope system, requiring a single drum, is used.

Main haulage. — Electric haulage seems more applicable to the endless-rope system (see fig. 225 on plate) than to the main and tail rope. It runs at a slow speed, about 2 to 3 miles an hour, almost continuously when working an endless rope, with a more or less steady load, whereas with the main and tail system the work is intermittent, the speed high, up to 12 miles an hour, and heavy strains are not infrequently thrown upon the gearing and motor.

There are several methods of transmitting power from the motors to the haulage gears, some being driven through worm or spur gearing, whilst others are driven by belt or ropes. There is an advantage in the latter method, because to a certain extent sudden strains or shocks caused by tubs being derailed are not felt so severely by the motors, as the belts take up a good deal of the shock. Owing to the damp and

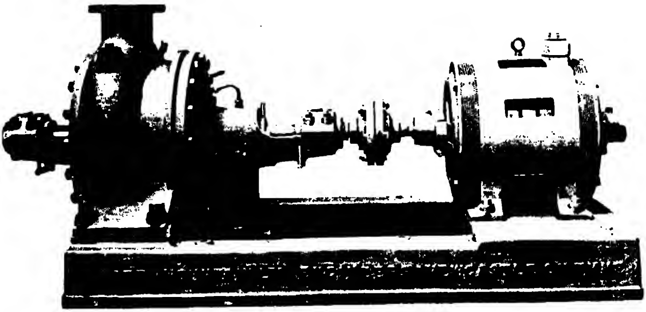


Fig. 224.—Three-stage Centrifugal Pump with Three-phase Motor

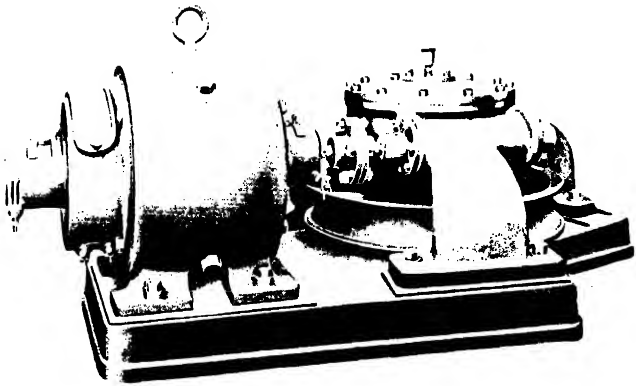


Fig. 225.—Endless-rope Gear with Direct-current Motor

dirt that is usually present underground, rope or belt transmission is rarely used, spur gear being preferred.

For controlling the motors for hauling, various types of starters and regulators are in use, some of the designs being ingenious. The liquid form is much used, as it admits of the load being put on gradually, avoiding jerks of the haulage ropes and chains.

Electric locomotive haulage is not considered an economical method in mines in this country. The locomotive itself is heavy, and absorbs a good deal of power when in motion, and the bare wires which are required overhead in the roadway underground are necessarily attended by considerable risk to workmen. This method is now prohibited in any part of coal mines. By a recent order (1920) locomotives carrying their own power are to be allowed in British mines. This means that where the gradients are flat, storage battery locomotives will in all probability be profitably employed.

In American and Continental mines electric locomotives are used in many instances. The locomotives collect the current from the overhead wire, and are run upon the main roads, or engine planes, where there is sufficient height.

Electric Winding in Shafts.—The first adaptation of electricity for winding materials at mines was in small shafts underground. Gradually larger electrically driven winders were built for winding to the surface, and now, both in England and other countries, large electric winding plants are in operation. An example of such a plant is shown in fig. 226 on plate.

In this country the earliest electric winding engines were erected at St. Hilda Colliery and Lumley Colliery, County Durham; and at the Great Western Colliery Company's Maritime Pit, South Wales. The current in each case is taken from an electric supply company, and no steam is generated or used at the two former collieries, all the machinery on the surface and underground being operated by electricity. By taking large

quantities of power from a supply company at a low rate, and avoiding the raising of steam, electric winding may be advantageous and economical.

Most of the winders are placed at one side of the shaft, with the ropes going over pulleys above the shaft, as is usual with steam winders. At some of the Continental mines the drums and motor are placed at a suitable height immediately above the shaft, so that the winding or hoisting ropes drive in a perpendicular direction from the drums. The only advantage there is in this arrangement is in the less wear and tear of the winding rope, as it is only bent once; whereas in the usual way it is bent the second time, and in the reverse way, in passing over the shaft pulley, but in the majority of cases this advantage is so slight as to be not worth while noting.

The disadvantage which appears to attach to electric winders being placed immediately over the shaft is that they might be rendered useless in the event of an explosion occurring in the mine and the force reaching the surface.

It does not appear likely that electric winding will be adopted generally at main shafts. The cost of installing is very heavy, and although electric winders may, in some instances, be more economically run than steam winding engines, the latter will always be a serious rival to electricity because they use the steam direct, whereas the electric winder requires the heat energy to be first converted into electrical energy, through the medium of a steam-engine, then from electric energy back to mechanical energy, through the medium of a motor, thus necessitating the loss due to two conversions in place of taking the heat energy direct, as the steam-engine does.

Where circumstances favour electrical winding, the winding gear can be constructed to give smooth steady running with absolute control, and as great reliability as can be obtained from a steam winder.

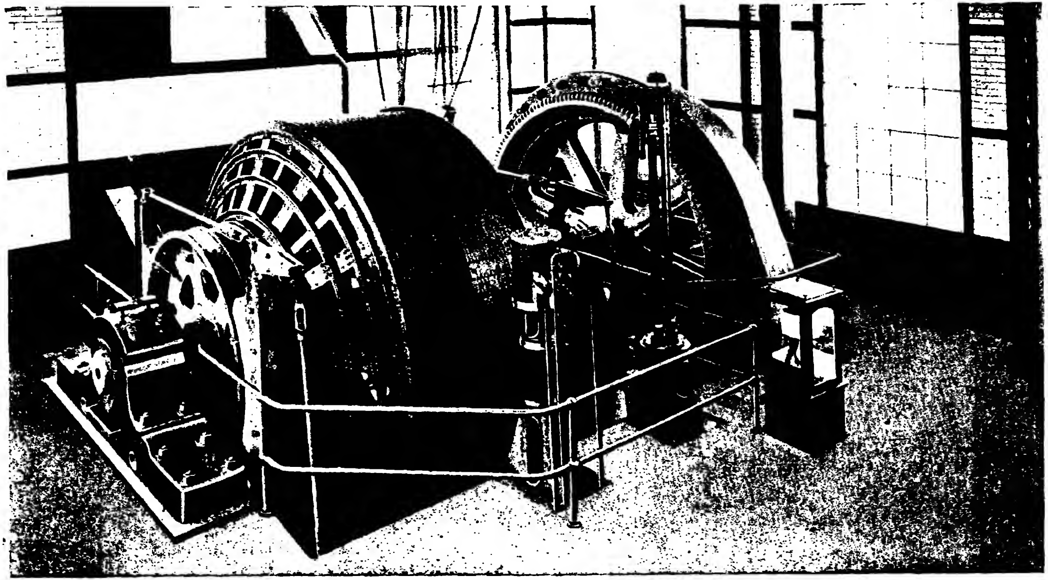


Fig. 226.—1500-h.p., 58-r.p.m. Westinghouse Electric Winder

CHAPTER XXXIV

SURFACE ARRANGEMENTS OF MINES

Construction and Arrangement of Pitbanks. Methods of Unloading Tubs. Screening, Picking, Washing, and Loading Coal.

Pitbank or Heapstead.—The surface arrangements of mines should be laid out so that the maximum output of coals anticipated may be dealt with expeditiously and satisfactorily. The winding engine should be large enough to wind with ease the daily quantity from the lowest seam, and all other arrangements and appliances should be made in proportion.

The coal tubs are rarely removed from the cage at the surface level; generally a stage, termed a pitbank or heapstead, is erected at a height of from 20 to 30 feet above the surface level, and this is made the place for the cages to stop at and tubs to be changed. The keps, which have been described already, are fixed at this level, and as the tubs are being changed, the cage rests upon the keps and is on a level with the landing-stage.

The staging in many cases is supported upon wood legs, which stand upon low masonry pillars. The staging or floor is of wood, and roofed for the protection of the workmen. Frequently iron columns are used, instead of wooden uprights, to support the floor, sides, and roof. In some cases the whole of the heapstead, including the pulley frames, is of iron.

The object of having a stage of 20 or 30 feet in height is to allow of the cleaning and sizing of the coal, and the discharging of it into railway wagons without the necessity of lifting it a second time. It is raised in the cage above the level of the railway wagons, so that it can be easily dropped into them.

The area and arrangement of the stage depends chiefly upon the output, and upon the processes through which

it is intended to put the coal before loading it into wagons. Where the output is large, room must be provided for the rapid emptying of a large number of tubs, and to accomplish this a sufficient number of screens must be put up in close proximity to the shaft, either on one or on both sides of the stage. If the railway is laid on one side only, then the screens must also be on that side only; but where a large daily quantity of coal must be dealt with, a better arrangement is to have screening on both sides.

There must be sufficient space on the stage to allow of one or more weighing machines being fixed, so that the exact weight of the mineral raised may be ascertained.

In a few cases workshops for the repairing of tubs, and a smithy for the sharpening of picks and other tools, are erected upon the stage, so as to be conveniently near the mine.

The stage must be arranged in such a manner that the tubs may be easily and quickly changed, and then emptied, when they reach the surface. As soon as the cage is lowered upon the keps, the catch is lifted, and the full tubs are pulled out of the cage, and at the same time empty tubs are pushed in from behind. The catch to keep the tubs in proper position in the cage is then put down, and on the necessary signals being given to the engineman to start the cages, the cage at the surface is slightly raised to allow the " banksman " to draw back the keps, and then it rapidly descends. This refers to a single-deck cage.

When the cage has two decks the bottom deck is changed first, then the cage is raised slightly, the keps are drawn back, and the cage is lowered until the top deck rests upon the keps. As soon as the tubs are changed, and the signals have been given, the keps are drawn back and the cage descends into the mine.

Sometimes when there are two or more decks a second stage or platform is arranged above the main one, so as to facilitate the changing of the tubs at the different

decks. When this is done at the surface, similar arrangements must be provided at the shaft bottom.

The top of the shaft is securely fenced around, except at the points of entrance to each cage. These points are protected during the ascent and descent of the cages by iron gates, which are free to slide in guides a few feet above the level of the stage. They are so arranged that they are lifted by the cage top on reaching the surface, and as the cage again leaves the surface they descend into their proper position and protect the shaft.

At all intermediate stopping-places, and at the shaft bottom, similar fence gates must be provided. Generally gates sliding horizontally are used at these points.

Tipplers, Tumblers, or Kick-ups.—The full tubs as they leave the cage are pushed forward to the weighing machines. Most stages are covered with iron plates, termed *flat-sheets*, upon which the tubs run easily, and upon which they may be pushed and turned in any direction. When weighed, the tubs are then pushed to the tumblers or kick-ups to be emptied upon the screens.

The work of discharging the coal out of the tubs on to the screen would be a slow and laborious process were it not for the contrivance, variously termed *tippler*, *tumbler*, and *kick-up*, which is fixed on the level of the stage at the top of each screen. This discharges the coals from the tub very quickly, and the tub is readily pushed in and drawn out. It consists of an iron frame working upon an axle at each side. Two rails are laid upon the bottom of the tippler, on to which the tub is run, and as soon as the tub has entered, the tippler turns over with it, and the coals are discharged on to the screen.

The common form of tippler turns over endways in a forward direction, and the coals are shot forward on to the screen. In others the tub is turned over sideways, and a third kind turns over endways in a backward direction. A very good tippler is one which turns over backwards, or towards the banksman, making at the same time a half revolution, the speed of which can

be regulated, so that the coals may be emptied upon the screen as slowly as the banksman may wish. The banksman has the tippler under his control by means of a brake attached to its side (see fig. 227, T). As soon as the tub is emptied the tippler is easily brought round to its original position, and the tub is then pulled out and pushed back to the shaft, to be returned to the mine.

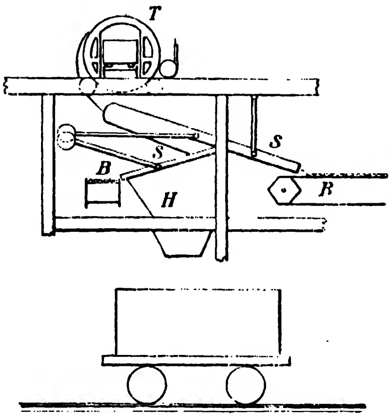


Fig. 227.—Banking Arrangements

T, Revolving tippler. S S, Jigging or shaking screens. H, Hopper. B B, Coal travelling belts.

Elaborate arrangements are made at some collieries for the transport of the tubs from the cage to the tipplers, and of the empty tubs from the tipplers back to the shaft. The object aimed at is to have the work done expeditiously by gravitation and machinery, with as little manual labour as possible. The distance from the shaft to the tipplers is laid with rails having sufficient fall to cause the tubs to run by the force of gravity; when emptied, they return along another set of rails having the necessary inclination in the opposite direction, and when they arrive at their proper places near the shaft they are hoisted to the required level by means of a steam hoist, or they may be raised by a "creeper".

Screening, Picking, and Loading Coals.—Screens are used for separating the various-sized pieces of coal received from the mine. In some mines only the large pieces of coal, termed *round coal*, are filled and sent to the surface, the small pieces being cast back and

left underground. In most cases, however, the coal is filled into the tubs in the condition in which it is got, round and small mixed together. For marketable purposes such coal is sized on reaching the surface, the largest pieces being separated from the rest, and sold as *round* or *best*, the smallest pieces, termed *small*, being sold at the lowest price. There may be several different-sized kinds between round and small, to which various terms are applied in different districts. As the round coal brings the highest price, it is a matter of great importance at most mines that a large percentage of the coal sent out be of this description.

The ordinary screen consists of a wood or iron spout, in which are fitted iron or steel bars. The angle of inclination of the screen is frequently about 20 degrees, depending, however, somewhat upon the condition of the coal to be screened. The bars are placed lengthwise down the screen, and are fitted into combs, which drop into slots or shoes bolted to the sides of the screen. The bars are made of wrought iron or steel, and are usually flat-topped, though in some cases they are round-topped. At the bottom of the screen is a table, upon which the coal rests before being shovelled into the wagon. As the coal slides down the screen to the table the *small* falls through the bars, only the *round* reaching the table. The round coal after reaching the table is cleaned, and all refuse separated and thrown on one side. The cleaning in some cases requires a large amount of labour. Inferior bands of coal or bands of stone may occur in the seam, and should be kept separate from the good coal. In some instances it may be possible to have the inferior coal filled separately, and the stone band thrown back and left underground; but thorough separation and cleaning are rarely carried out in the mine, and it is therefore necessary to arrange for this to be done more effectively at the surface. After the coal is cleaned it is shovelled off the table into the wagon standing on the railway below. In the case of soft coal much breakage

may result from the fall into the wagon, and to avoid this the coal is sometimes lowered into the wagon in buckets. The small coal which passes through the screen bars falls into a hopper, from which it may be loaded direct into wagons, or, if needful, be first subjected to further sizing and cleaning.

The railways under screens are generally arranged with a slight inclination, so that the empty wagons will of themselves run to the screens, and when filled run from under them. To avoid accidents, a clear space between the wagons and the uprights should be left, so that a man may pass between them. It is also a very good plan to pave with bricks the railway under the screens, so that any coals dropping may be cleaned up again without getting mixed with any dirt.

Jigger Screens, Travelling Belts, Revolving Tables.—Coal from seams containing inferior coal and stone is very frequently inefficiently cleaned upon the ordinary form of screen, especially where a large quantity of coal must be dealt with in a short time. And coal inefficiently cleaned is of reduced market value, and may be altogether declined by buyers owing to the many impurities mixed with it. Various appliances have therefore been introduced for the more satisfactory separation of the small coal and dirt, some of which are largely in use. These are the *jigger* or *shaking screens*, *travelling bands* or *belts*, and *revolving tables*.

Jigger or Shaking Screen.—This is a steel-wire screen to which a backward-and-forward or shaking motion is given by a pair of eccentric sheaves working on a shaft from an engine usually placed under the heapstead. The throw of the eccentrics does not exceed a few inches. When the coals are emptied by the tippler upon the screen, the shaking motion moves the coals about, and the small falls through the wires. The screen is fixed at an angle of from 10 degrees to 18 degrees, and is worked at the rate of from 70 to 100 strokes per minute, the coals gradually moving forward upon it until they

are shaken off into a spout leading to the wagon, or on to a travelling belt for further treatment. Shaking screens of wire are generally used, but sometimes bars or perforated plates are adopted.

Travelling Belts or Bands.—These are made endless, and travel around a drum fixed at each point where the belts return. The motion to the belts is imparted by the drums, which are driven at a slow rate by bevel gearing. The belts are from 20 feet to 60 feet in length, and from 3 feet to 4 feet in width, according to the requirements. They are made of iron or steel plates about 9 inches broad, and are attached to long-linked chains. The sides of the drums must correspond to the breadth of the plates, so that there may be no breaking or bending as the belts travel around. The belts travel at from 30 feet to 60 feet per minute, and the coals as they fall from the screens on to the belt are carried forward. Men and boys stationed on platforms on each side of the belt then examine the coals as they pass along, and pick out any stones or coarse pieces of coal. For effective cleaning, the coals should be thinly and evenly distributed upon the belt.

Revolving Tables.—These are circular tables made to revolve by gearing from an engine. They receive the coals at the bottom of ordinary screens, and, in revolving, the refuse is taken out and the coal assorted as required by men and boys. They travel very slowly, and curved scrapers are fixed so as to sweep off the coal after it has been once revolved. Sometimes they are perforated like a screen or riddle to allow the small to pass through, and thus secure still more efficient screening. Some of the advantages claimed for these tables are:

1. They enable dirt to be picked out better than by any other method.

2. They assist the screening operations, and enable different kinds and sizes of coal to be assorted, whereas travelling belts do not assist this work.

3. They are not liable to get out of order.

Coal Washing.—Nearly all coals to be used in the manufacture of coke contain impurities, such as stones and brasses, and unless these are removed the coke contains the impurities in the form of ash. The coal-cleaning methods already described may be satisfactory for dealing with coal from 3 inches square and upwards, but smaller coal, particularly of the fineness necessary for coking, cannot be hand-picked efficiently, owing to the smallness and quantity of the stones to be removed. In order to satisfactorily remove the stony materials from the coal, a process of washing has to be resorted to.

There are numerous designs of coal washers, but the principle upon which their efficiency depends is the same in each case, namely, the difference of the specific gravities of the stones and coal. The separation takes place by the heavier materials falling to the bottom of the washing machine, while the lighter material, which is the coal, is carried away by the stream of water and delivered where required.

In recent years much attention has been given to the subject of washing, owing to the opening out of seams containing more stones, and to the demands of coke users for coke containing a low percentage of ash.

The chief objects aimed at in the construction of a washer are:

1. To remove as efficiently as possible all stone and other impurities in the coal.
2. To avoid loss of coal in the process of washing.
3. Minimum capital expenditure and low washing costs.

The numerous washers in use may be classified as follows:

1. The trough washers.
2. The jigging washers.
3. The table washers.

1. The earliest known washers were made on the trough system, and improved makes are still in use.

The ones mostly used at present are the Elliott, Burnett's Patent Coal Washer, and the Blackett. The Elliott consists of metal troughs in which are slow-moving scrapers drawn by a chain which moves them forward against the stream of water and coal. The stones are caught by the scrapers and conveyed to the upper end of the trough and discharged, while the coal is carried to the other end by the water. In the Burnett Washer the trough with dirt stops moves upwards against the water and coals. The dirt is caught at the stops, and the coal is carried down by the water. The Blackett is an inclined revolving tube with a continuous spiral inside. The water carries the coal downwards, and the spiral carries the dirt up the inclined tube and discharges it at the top.

2. Of the jigging type there are some efficient machines of very large construction, such as the Humboldt, the Luhrig, and the Baum. In these different makes of machines there is a similarity in the general arrangement, a pulsating action being produced in a trough containing water and unwashed coal, which agitates the water and lifts the coal upwards, and the stones fall to the bottom. Some very large plants of these makes are at work in various parts of the country to prepare coals for coking in patent coke-ovens. Many of them are capable of washing 1000 tons of coal per day. The Robinson Washer, which was largely used some years ago, is a modification of this type. There is no pulsation, but a rotating action is given to a stream of water in a wrought-iron inverted cone. The coal is carried round by the flow of water, and as it rises to the top it overflows at one side, and the stones fall to the bottom of the cone, from which they are discharged.

3. The Craig Washer is probably the best example of the table type, and it is in successful operation at Thornhill Colliery, Yorkshire, for washing very small coal. It consists of a table, wide at one end, where the water and coals to be washed drop on to it, and narrow

at the other end, where the coals and water run off. A back-and-forward movement, with a jerk at the end of each movement, is imparted to the table, in order to carry the coal forward and the stones backward and so separate them, and the stones fall from the table at the corners. One table can deal with 100 tons per day.

CHAPTER XXXV

ACCIDENTS IN MINES

Firedamp and Coal-dust. Falls of Roofs and Sides

Death-rates from Accidents.—The tables given on page 361 are prepared from the Statistics of Mines issued by the Home Office. They show the ratios of lives lost to the number employed in and about mines in the periods between the various Coal Mines Regulation Acts since the Act of 1850.

The tables show that the occupation of the miner is very much safer now than in 1850, there being a reduction in the proportion of fatal accidents to the number of employees. Under the first Act the average ratio was 1 fatal accident to 223 persons employed, whereas for the year 1908 there was 1 fatal accident to 756 persons employed. In 1915 the ratio was 1 fatal accident to 736 persons employed, whereas in 1918 the ratio was 1 fatal accident to 720 persons employed, so that the improvement is not being maintained, but instead the situation seems to be turning worse.

This increase of safety in the conditions of employment at mines is not entirely due to stringent legislative enactments. Better appliances, increased knowledge, with improved methods of working have also had a share in rendering mining operations safer. But we now seem to have reached a point where improvement

has ceased, and it is an open question if many of the legislative fads embodied in the latest Mines Act have not led to more accidents rather than to lessening them. A careful analysis of the whole position points to the conclusion that we have now arrived at a position where we should legislate less and educate more, as it is impossible to instil foresight and discrimination by Act of Parliament.

COAL MINES REGULATION ACT.—DEATHS

Ratios of deaths by accidents to the numbers of persons employed in and about mines under the former and present Coal Mines Acts.

Year.	Deaths caused by Accidents.						Persons Employed.	
	Firedamp Explosions.	Roof and Sides Falling.	Shaft.	Miscellaneous Underground.	Miscellaneous Surface.	Total.	Above Ground and below Ground.	Ratio of Persons Employed to each Death.
Act for inspection of coal mines in Great Britain, 1850								
Totals, 5 yrs., 1851–55	1155	1842	1183	509	238	4927	1,147,341	
Averages	231	368	236	102	48	985	229,468	233
Act for inspection of coal mines in Great Britain, 1855								
Totals, 5 yrs., 1856–60	1286	1925	936	677	267	5091	1,312,981	
Averages	257	385	187	136	53	1018	262,596	258
Act for the regulation and inspection of coal mines and ironstone mines connected with coal, 1860								
Totals, 12 yrs., 1861–72	2690	5054	1783	2309	925	12,761	3,981,374	
Averages	224	421	149	192	77	1,063	331,781	312
Coal Mines Regulation Act, 1872, applying to coal, stratified ironstone, shale, and fire-clay mines								
Totals, 15 yrs., 1873–87	3447	6856	1734	2986	1365	16,388	7,634,520	
Averages	230	457	115	199	91	1,092	508,968	466
Coal Mines Regulation Act, 1887, applying to coal, stratified ironstone, shale, and fire-clay mines								
1888	49	471	75	209	84	888	534,945	602
1911	36	617	69	414	129	1265	1,067,213	843
Coal Mines Act, 1911								
1914	26	596	66	398	133	1219	1,133,746	930
1915	41	673	61	392	130	1297	953,642	736
1918	160	685	47	385	124	1401	1,008,867	720

The Tables also show that the two chief sources of accidents are: 1. Explosions; 2. Falls of Roof and Sides. It is an unfortunate fact that the greatest number of accidents are due to falls, and far exceed those from explosions; in fact, nearly one-half of the total fatal accidents are due to falls. The accidents are numerous, and although the deaths resulting may not exceed one or two in each case, yet the total at the end of each year is large. Fortunately, explosions are rare, but on their occurrence the number of lives lost is usually large.

For the year 1918 the percentages of deaths at collieries from various causes were as follows:

Explosions of firedamp or coal dust ..	11·42	per cent.
Falls of ground	48·89	„
Shaft accidents	3·35	„
Miscellaneous underground	27·48	„
Surface	8·86	„
	100·00	

Explosions.—Explosions in mines may be caused by the presence of (1) Firedamp, (2) Firedamp and Coal-dust together, and (3) Coal-dust.

(1) *Firedamp Explosions.*—Many explosions which have occurred have been proved to be pure firedamp explosions. In mines where the seams worked give off firedamp, the atmosphere at the face of the freshly-cut coal, and in old working places, goaves, and returns, will almost always contain some proportion of firedamp. This proportion may be too small to be detected by the ordinary method, namely, observing the appearance of the flame of the ordinary safety-lamp, which becomes enlarged or elongated, or topped by a *blue cap* or *aureole*, when there is 1½ per cent or more of firedamp present in the atmosphere. Workings examined with safety-lamps are declared to be free from the presence of firedamp when the flame shows no " cap ". The air, unless carefully tested for firedamp, may contain any percentage below 3, or, if skilfully tested, below 1½ per cent, and be

reported free from gas. Such a mixture is not in itself dangerous, but a slight further addition of firedamp may quickly produce an explosive mixture. Gas exudes constantly in more or less regular quantities from the coal, goaves, and fissures and cavities in the roof and floor; the quantity, however, may be increased by a sudden reduction of atmospheric pressure even before such reduction is indicated by the barometer, and the atmosphere in the mine for a certain area thus be rendered explosive. Blowers and sudden outbursts of gas from the coal or the adjoining beds sometimes occur; and the large volumes given off in a short time may speedily give rise to an explosive mixture filling large areas. Many disastrous explosions that have occured in this and other countries in well-ventilated mines, some of which were inspected and declared free from firedamp shortly before the explosions, have, it is thought, been due to sudden outbursts of gas.

(2) *Firedamp and Coal-dust together.*—During the last fifteen years the subject of coal-dust and the part it plays in mine explosions has been much investigated. Coal-dust was first suggested as being an element of danger in coal-mines in the year 1803, after the occurrence of a fatal explosion at Wallsend Colliery, Northumberland; but it was not considered by many to be an agent in aggravating and extending explosions until after the occurrence of several violent explosions, which appeared to originate at points in the mines where firedamp had not been detected, and where it was considered to be almost impossible that it could be present in quantities sufficient to produce such disastrous results. At these points shot-firing had been practised, and coal-dust was known to exist there and along the roads where the explosion travelled.

A very large number of coal-dust experiments have been carried out, and they go to show that a small proportion of firedamp present in air in mines along with dry coal-dust may cause an explosion if ignited. The

Royal Commission appointed to inquire into Accidents in Mines investigated the subject of coal-dust, and in their final report say: (1) " The occurrence of a blown-out shot in working places where very highly inflammable coal-dust exists in great abundance may, even in the total absence of firedamp, possibly give rise to violent explosions, or may at any rate be followed by the propagation of flame through very considerable areas, and even by the communication of flame to distant parts of the workings where explosive gas-mixtures, or dust deposits in association with non-explosive gas-mixtures, exist.

(2) " The occurrence of a blown-out shot in localities where only small proportions of firedamp exist in the air, in the presence of even comparatively slightly inflammable, or actually non-inflammable but very fine, dry, and porous dusts, may give rise to explosions, the flame from which may reach to distant localities, where either gas-accumulations or deposits of inflammable coal-dust may be inflamed, and may extend the disastrous results to other regions."

There is not any mention of coal-dust in the Coal Mines Acts until, as an outcome of the investigations and reports of the Royal Commission, rules were inserted in the Act of 1887 referring to the firing of shots in mines where dry coal-dust exists.

(3) *Coal-dust Explosions.*—Experiments which have been made with various coal-dusts go to show that certain kinds of very dry, finely·divided, and highly inflammable coal-dusts may be ignited by an explosive charge and cause a violent explosion without the presence of any firedamp. The investigations into some recent colliery explosions have also shown that coal-dust distributed along dry haulage roads may be ignited by the firing of a shot where the dust is present, and give rise to an explosion the violence of which may extend and increase in force as it traverses the main roads along which the dust is distributed. It has also been pointed out that a much smaller quantity of dust is sufficient to

extend and carry on an explosion than was generally expected. The most disastrous of the colliery explosions of recent years have occurred on the main intakes and haulage roads, and while on the one hand the force of the explosion has rarely reached the face of the workings, yet on the other hand it has in all cases reached the shaft bottom, against the ingoing current of fresh air, with more or less violence.

Mr. J. B. Atkinson, H.M. Inspector of Mines, in his report on the circumstances attending an explosion at Washington Colliery, Durham, February 20, 1908, says: "I do not think that an excess of dust has much effect on the extension of violence or an explosion after it is once established; so long as there is sufficient fine dust capable of being raised as a cloud and enough to combine with the oxygen of the air, which is always a limiting factor, the part the excess plays is doubtful. An excess of dust ensures that there is sufficient of the finer particles, and probably has considerable influence on the initiation of an explosion, whether due to dust alone or dust and firedamp in combination; but, after an explosion is fully established, my experience is that the excess of dust (unlike an excess of firedamp) neither increases nor retards the explosion."

Treatment of Coal-dust.—The successful treatment or disposal of coal-dust has been found to be an extremely difficult matter.

The Coal Mines Act, 1911, requires:

"The floor, roof, and sides of the roads shall be systematically cleared so as to prevent, as far as practicable, coal-dust accumulating."

Such systematic steps, either by way of watering or otherwise, shall be taken to prevent explosions of coal-dust occurring or being carried along the roads.

No shot shall be fired unless the shot-firer has examined the floor, roof, and sides of all contiguous accessible places, within a radius of 5 yards of the place where the shot is to be fired, for coal-dust, and has taken

sufficient steps to render any dust within that area harmless. The Explosives Order, 1912, imposes further shot-firing restrictions.

Removing Dust.—To clear the floor, roof, and sides of the roads, it is usual to fill the coal-dust from the floor into tubs, and send it out of the mine. The roof and sides are the more difficult to deal with. Brushing is sometimes resorted to, but the dust is so impalpable and dry that the air current carries forward the lightest particles and deposits them elsewhere.

Watering.—Very often the floor of the main roads is watered, but in many mines difficulties arise owing to floor movement due to the water, and it is as important to damp the dust adhering to the roof, sides, and timber, and no satisfactory method has been arrived at for accomplishing this.

In many cases pipes containing water under pressure are laid along the main roads. At intervals branches are affixed with sprayers to break up the water into fine spray, which is carried forward as moisture by the air, and damps the roadway, top, sides, and bottom, but very often the dampness varies in degree, or may be quickly evaporated.

Another method is to employ a special tub filled with water, and by means of a small pump or pumps, connected directly to a small crank formed upon one axle of the tub, the water is forced through an air-vessel, and from thence one or more nozzles or sprayers produce fine spray.

Stone-dusting.—For many years experiments have been made, and in recent years a Government Committee on Explosions in Mines have been testing and reporting on the use of incombustible dust as a means of retarding, if not entirely preventing, coal-dust explosions. The conclusion drawn from the experiments is that incombustible stone-dust, in suspension in weak mixtures of firedamp and air, renders such mixtures less explosive, and that stone-dust artificially mixed with coal-dust in

the proportion of 1 to 1 renders the coal-dust inexplosive.

This method of treating coal-dust with a view to reducing the risk of explosion in dusty mines is in operation at a large number of collieries, and is now legally enforced by a recent order under the Mines Act.

The inert dust may be flue-dust collected from boiler flues, or specially prepared by grinding soft shale free from silica in a mortar mill.

The coal-dust is first cleaned away from the road where it is intended to apply the stone-dust. The latter is distributed by throwing it on the sides and roof timbers until they have a whitish appearance. The application is commenced at the outbye end of the intakes, and in very hot and dry dusty mines, where it is considered advisable to stone-dust part of the returns, it is done from the inbye end. It has been found that the quantity of stone-dust required on an ordinary sized main road varies from ½ cwt. to 2 cwt. per lineal yard. The quantity depends upon the conditions of the mine, such as the dryness, and the pureness of the coal-dust, consequently it is necessary to have analyses made of the dust in the mine before the application of stone-dust, and at intervals afterwards. The making of such analyses is now rendered compulsory by a Home Office Order already referred to.

Falls of Roof and Sides.—As already stated, the largest number of accidents in mines are due to falls of stone from the roof and sides of the roadways and working places. The character of the roof and floor of coal-seams and the inclination of the beds vary so much in different seams, and even in the same seam, that it is very difficult to lay down fixed systems of timbering suitable for all places. Stone forming the roof may undergo various changes from pressure and movements of the strata after the roadways have been made; it may become full of fractures, breakers, and slips; it may bend down and break the timber; or it may fall between and over the timber.

It is rare, however, for a fall of roof to involve more than one or two men, as the falls occur at the faces where the men work singly or in pairs. Collapse of roof over a large area seldom occurs, and when it does it is usually preceded by sufficient warning to enable the men to get away. It is the stones which, when the coal is extracted, drop unexpectedly from vertical undetected slips and joints, and slippery horizontal partings, that cause the most of the face accidents.

Systematic timbering, requiring supports at stated intervals, was introduced some years ago, and the Coal Mines Act, 1911, has laid down rules necessitating the fixing of distances for supports, and plans to indicate the proper method of setting timbers, at each mine.

In the North of England, officials termed *deputies* have charge of a certain number of working places, and are responsible for the proper timbering of these as the coal is extracted. They also prepare timber for each place, and must see that it is suitable for use, in case it is necessary for the coal hewer to " set " timber during the intervals between the deputy's visits. In other coal-fields the face timbering is attended to by the hewers, the officials inspecting and directing them in the work.

Many accidents occur in the drawing or recovery of timber. The work is generally entrusted to men of experience, but sometimes a sudden overweighting occurs, extending and breaking down the timber where the men are working.

Haulage Accidents.—A considerable number of accidents occur on the roadways to persons whose duties involve the handling of the tubs and appliances for conveying the coal from the face to the shafts. Near the face the tubs are moved by hand, and where the gradients are steep, runaway tubs are often the cause of accidents. Also, where the tubs are moved by ponies, the persons in charge sometimes fall in front of the tubs, or come in contact either with the roof or sides when riding upon the tubs, and so receive injury. Riding on limmers is

a fruitful cause of accident where these appliances are used.

On self-acting inclines accidents occur through runaway tubs, owing to stopchocks not being in position, ropes or coupling chains not being attached, and sometimes to broken ropes.

On engine planes where the tubs are moved mechanically, many accidents occur through persons inadvertently stepping in front of the tubs, or failing to step into a refuge on the approach of the train of tubs. On haulage roads where the gradients are steep, accidents are caused by runaway tubs from drawn sockets or broken couplings, chains, and ropes.

Measures of a precautionary kind against these occurrences are enjoined in the Mines Act, requiring travelling roads, refuge holes, &c., on haulage roads.

Shaft Accidents.—Many of these occur when shafts are being sunk, from causes such as inrushes of sand and water, slipping away of shaft sides before the permanent supports have been put in, overwinding of kibbles, and falling material. In ordinary winding shafts accidents occur from overwinding, breakage of ropes or chains, misunderstandings of signals, and falls from surface or openings in the shaft.

For the avoidance of shaft accidents it is essential to have properly equipped engines with adequate brake power, and automatic contrivances to operate in the event of an overwind. The winding ropes should be of good manufacture, and must be tested with seven times more than the ordinary load to be lifted. In addition to the daily external examination of the ropes, an internal one should be made periodically when "re-capping" or "re-socketing". Cages, signals, indicators, and keps should be maintained in an efficient condition, and all openings or shaft "insets", including top and bottom, should be fitted with suitable gates.

Miscellaneous Accidents.—A number of accidents

arise during the operation of blasting. The safety regulations in connection with the use of explosives have already been enumerated. Others occur through suffocation by natural gas, the breaking out of underground fires, inundations of water, and from the use of machinery and various other causes which from time to time claim their toll of victims.

SURFACE ACCIDENTS occur on the surface in connection with coal-cleaning and loading apparatus. In moving trucks at screens men are apt to become careless, and get caught between buffers. Care should be taken when erecting the screens to leave sufficient clearance for the passage of workmen between the upright columns and the trucks. On the colliery railway sidings, coke ovens, and in the repairing shops, accidents are not infrequent.

ELECTRIC ACCIDENTS.—Owing to the extended use of electricity in mines during recent years a new source has been introduced from which accidents may arise. These may consist of:

1. Shock to workmen.
2. Setting fire to combustible material.
3. Ignition of firedamp.
4. Electric blasting operations.

1. SHOCK TO WORKMEN.—Owing to the varying degrees of susceptibility to shock possessed by different people it is impossible to fix upon a pressure which would be safe under all circumstances and cases. The damp atmosphere of mines tends to increase the shock. One person might be affected by a shock from mains carrying a pressure of 100 volts, and another person might not be affected by several hundreds. The Electricity Committee considered a " low pressure supply " one of 250 volts; a " medium pressure supply " one from 250 to 650 volts; a " high pressure supply " one from 650 to 3000 volts.

2. SETTING FIRE TO COMBUSTIBLE MATERIAL.—Fire may be caused by (1) short circuits, that is, a positive

conductor coming into contact with a negative conductor, causing a heavy flow of electric current; (2) bad contacts, causing arc-ing of the current. Many fires have occurred in and about mines owing to one or the other of these causes, but if reasonable care and forethought were exercised in the installation and maintenance of the plant these should not occur.

3. IGNITION OF FIREDAMP OR COAL-DUST.—The adoption of electricity in mines has introduced an element of danger unless properly safeguarded. With electric cables and motors for pumping, hauling, or coal-cutting in the workings of " gassy " mines there is a risk of sparks by short-circuiting or bad contacts from the cables, switches, or commutators. Such risk, however, may be reduced, if not altogether eliminated, by the use of materials of the best quality, properly installed, by having motors in flame-tight cases, and by careful observance of the Electric Special Rules.

4. ELECTRIC BLASTING OPERATIONS.—Numerous accidents have occurred, and may arise from:

1. *Premature firing of the Shot.*—Almost all the accidents due to electric firing have been traced to this cause, and usually arise owing to some neglect or misunderstanding between the shot-firer and the workmen in the vicinity. If the firing cable be first connected to the detonator wires at the shot-hole, and not attached to the terminals of the exploder, accidents of this nature should not occur. In the case of missfire, the exploder should be at once disconnected.

2. *The ignition of Firedamp from a Spark, either in the Exploder or between the bare leads.*—Accidents from this cause are rare, and should not occur at all. High-tension magneto machines should be enclosed in a flame-tight case, and the detonator wires connected so as to avoid the possibility of a spark passing from one to the other.

UNDERGROUND FIRES.—Fires underground have frequently resulted in large loss of life. When these occur on main intakes, as has frequently happened, the men

beyond the fire are cut off from escape by the usual ways, either intake or return, and the smoke and fumes, often containing carbon monoxide, are carried into the workings.

Fires have occurred from carelessness in the use of naked lights, from friction of ropes and sheaves on haulage roads, from explosions, and from electricity.

Fires arising spontaneously in goafs and abandoned workings occur frequently in some coal-fields, and serious explosions of gas have been caused by such fires. The causes which give rise to spontaneous combustion, and the best methods to prevent their occurrence and to deal with them when they do occur, are matters still requiring investigation to determine them.

MINERS' DISEASES

Ankylostomiasis.—This is a contagious disease brought about by the agency of worms, which find their way into the bowels of the infected person. Cases have occurred in Cornwall metalliferous mines, but, so far, none have been known in the coal-mines in this country.

Nystagmus.—This disease among miners has increased very much in recent years. It consists of an involuntary oscillation of the eyeballs, and it is sometimes accompanied by other symptoms.

The chief causes are stated to be (*a*) the position in which a miner works at the coal face, (*b*) the small light from the safety lamp. Some medical authorities give the former as the sole cause, and some the latter.

There are doubtless contributory causes, some constitutional, and some mining.

Many medical men think that improved lighting will in time do away with the trouble. The only prospect of curing nystagmus is for the afflicted miner to discontinue work in the mine for a time,

AMBULANCE WORK

The Coal Mines Act, 1911, requires provision to be made at all mines for the supply and maintenance of ambulance appliances and the training of men in ambulance work.

Owing to the isolation of many collieries, owners in many instances have provided, in addition to the above requirements, motor ambulances of the most approved type, in order to reduce to a minimum the sufferings of injured persons when being removed to their homes or hospitals.

In some mines ambulance stations are placed at the junction of main roads; and an ambulance house must now be provided on the surface at all collieries. These are provided with stretchers, splints, material for dressing wounds, and rugs. In case of an accident at the coal-face, men are dispatched for a stretcher and ambulance material, and others attend to the patient. If seriously injured, messages are sent outbye to have the roadways cleared of tubs, and to have the ambulance carriage ready at the surface, and in many cases the medical attendant is apprised of the occurrence by telephone.

A good deal of impetus has been given by the Home Office and H.M. Inspectors of Mines to the formation of classes for colliery officials and workmen to receive ambulance instruction, or " first aid to the injured ", as formulated by the St. John Ambulance Association. The objects of the Association include: (*a*) The instruction of persons in rendering first aid in case of accidents; (*b*) the manufacture and distribution of ambulance material; and (*c*) the formation of ambulance depots at mines and other centres of industry.

The acquisition of a knowledge of ambulance work by miners, employed as they are in remote places underground, in many cases 3 miles from the shaft, is of great advantage.

The chief lessons to be learnt by miners are:

1. The lifting of injured persons and the carrying of the stretcher.

2. The proper use of splints and bandages upon fractured bones, to prevent simple fractures from becoming compound.

3. To arrest hæmorrhage and dress wounds.

4. To understand the effect produced by the inhalation of the various gases that may be present in coal mines, and how to treat persons that have been overcome by them.

5. The effects produced by electric shock; how to handle the injured and avoid receiving shock also; and the method to adopt to restore the apparently killed.

RESCUE STATIONS

The provision of rescue stations, equipped with suitable apparatus and trained men, has been adopted in the mining districts of this country, and in America, France, and other parts of the Continent of Europe.

The object is to train rescue parties to penetrate some distance into the workings of a mine, after an explosion, to examine the state of the ventilation, and, where deranged, to restore it by building stoppings, clearing falls, &c.; also to enable men to investigate gob-fires, and operate in the work of extinguishing, digging out, or damming off underground fires, and generally to assist in the work of reclaiming and saving life and property.

The rescue apparatus may consist of one or other of two types:

1. Apparatus of the "diving-dress" type, in which a stream of fresh air is continually fed into a helmet by a hose from a compressor.

2. Self-contained apparatus — the appliances and chemicals for the production of a constant supply of breathable air are carried by the wearer.

The apparatus of the first-mentioned type is not

suitable in the case of explosions, but may be suitable for operations in the case of fires. The second type is suitable for rescue and recovery work at distances underground.

There are several makes of rescue apparatus, of which the following are the most prominent: The Fleuss, the Draeger, the Weg, the Tissot, the Meco, and the Aerolith. No particular make of apparatus to be adopted by each mining district has been specified by the Home Office, the selection being left to the mine-owners. Several rescue stations have been built and equipped with one or other of the different makes. Where the Aerolith has been installed the equipment consists of: one air-liquefying plant complete for manufacturing liquid air; ten "Aerolith" patent liquid-air self-contained mining rescue apparatus; receptacles of pure nickel, including all tubing; new improved face-mask helmets allowing natural breathing; and liquid-air containers in suitable cases for storage of liquid air.

Properly trained brigades, the number required being in proportion to the number of workmen underground, must be maintained at each colliery. Constant practice is necessary with the apparatus, in irrespirable atmospheres if possible, in order to ensure both men and apparatus being efficient, otherwise there is serious risk to the rescuers when they put on the apparatus and enter a dangerous atmosphere. Fatal results have attended the use of rescue appliances, due, probably, to defects in the apparatus allowing leakage, and in some cases preventing a proper supply of oxygen to the wearer.

Rescue and Aid Regulations of May 19, 1914, and further Regulations in 1920, have been issued by the Home Office, and should be referred to for particulars of the schemes for the training and practice of brigades.

CHAPTER XXXVI

SURVEYS AND PLANS OF MINES

Use of Magnetic Dial and Theodolite Underground. Reduction and Plotting of Surveys. Levelling Underground

Plans required at Mines.—The Coal Mines Act requires that there shall be kept in the office at the mine an accurate plan of the workings of the mine, showing the workings up to a date not more than three months previously, and the general direction and rate of dip of the strata, together with a section of the strata sunk through, or if that be not reasonably practicable, a statement of the depth of the shaft with a section of the seam. Every such plan must be on a scale of not less than that of the Ordnance Survey of 40 inches to the mile, or on the same scale as the plan for the time being in use at the mine. A plan of each seam worked is usually made on strong linen-backed paper, and tracings from it prepared on tracing paper for the daily use of the mine officials.

What is a Plan?—A plan is a diagram in miniature of the workings of a mine, and represents what could be seen by the naked eye looking down upon the mine from a point above (see figs. 103, 111). A plan should be accurate and complete, and show not only the position and progress of the working-places, but the irregularities and interruptions encountered in the seam, the intakes and returns, the engine-planes, wagon-ways, landings and sidings, and the goaves. The boundaries of the various royalties, or areas of coal to be worked belonging to different land and royalty owners, should be accurately surveyed on the surface and plotted on the plan. All roads, rivers, railways, and houses on the royalties should also be surveyed, and represented on each plan in their exact positions and relative proportions.

In making a plan, the first thing is to accurately fix on it the position of the downcast and upcast shafts in their true relation to the boundaries and other physical features on the surface. If these are erroneously located, the result will be an inaccurate map or plan of the workings in relation to the surface.

The meridian or north line fixed upon the plan may be either the true meridian or the magnetic meridian. If the true north line is used, all the bearings in surveys with the magnetic needle must be reduced to true north bearings before they can be plotted, because the true meridian and the magnetic meridian do not coincide. When the magnetic meridian is fixed upon the plan, the date of it is noted. As the magnetic needle is subject to changes of variation, in order to keep the plan correct the amount of such changes must be noted and the bearings taken in the surveys reduced to the plan meridian, or the latter must be altered in the same direction and to the same amount as the declination of the magnetic needle.

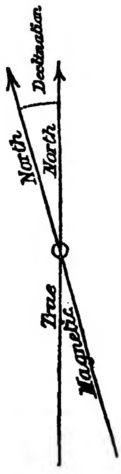


Fig. 228.—Illustration showing Magnetic and True North Lines

Declination (see fig. 228).—The amount of the declination or variation from true north of the magnetic needle is continually changing. This change is found to be of a tolerably regular increase or decrease, at one time moving to the west and at another time moving to the east of true north, and in passing in either direction it occupies a long and indefinite period. The variation at present is to the west of true north, and it is gradually decreasing in amount at the rate of about 8 minutes per annum. The greatest angle was reached in 1815, when it stood at 24° 27′ west of true north; since that date it has been travelling slowly back, and it is now (October, 1920)

14° 28′ west of true north at Greenwich, and decreasing at the rate of about 8′ annually.

Old Workings.—If any old workings are supposed to exist in the district they should be surveyed, if it be possible to enter them; if not, all plans or information relating to them that can be had should be secured, so that the old places may be shown on the colliery plans in their exact position. When workings are copied from old plans to be traced upon new plans, it is necessary to ascertain the amount of variation of the meridian since the date of the old workings, and in putting such workings on to new plans the meridian must be altered to an amount equivalent to the amount of variation, otherwise the relative position of the old workings to the new workings will be inaccurate.

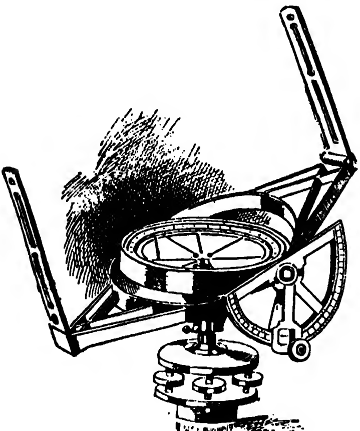


Fig. 229.—Circumferentor

Instruments used in Surveying.—Angular surveying in mines is done either with the *compass* or the *theodolite*. The measurement of the lines surveyed is accomplished by means of a small linked chain. The chain commonly used for surface and underground surveys in coal-mining districts is *Gunter's chain.* It consists of 100 links, each 7·92 inches long, the total length of the chain being 66 feet.

The compass, sometimes termed the *miners' dial*, or *circumferentor* (see fig. 229), is much used underground. It is a very valuable instrument, and from the simplicity of its parts, and the rapidity with which it can be

manipulated, it is very convenient and suitable for use in low, narrow, and cramped situations in underground workings. Many of the older forms of compasses still in use may not give precisely accurate results, but some of the modern improved dials are fitted up with minuteness, and give results as exact as those of a theodolite.

The essential element of a compass is the magnetic needle, which rests upon a steel pivot in the centre of the dial, whose circumference is divided into 360 degrees. The needle is a strongly magnetized bar of steel, in the centre of which is inserted a cap of agate, and this rests upon the steel pivot, so that the needle may turn freely and with the least friction, and so come to rest in the magnetic meridian. The north end of the needle is marked for the sake of distinction, and when first unlocked moves freely backwards and forwards before coming to rest. The dial is fitted with sights, levels, and a vernier, and like other instruments is screwed upon a three-legged stand. The legs are shod with iron, and are jointed at the top so that they can be kept together or moved in an outward direction.

The compass may be used to survey with (1) the needle loose and free to oscillate in the dial, and (2) with the needle fast or fixed.

The first of these methods is termed *loose-needle* surveying, and is mostly adopted for ordinary purposes. To make a correct survey all iron must be removed to a distance from the compass, or the needle will be attracted and deviated from its proper position and the bearings consequently be incorrect. The *bearing*, or the direction of a line or underground excavation, is the angle which it makes with the magnetic needle. The sights of the compass are directed along the line to a staff or a light, and when the needle comes to rest the number of degrees of the angle formed by the line of sight and the needle are read off the dial. This is termed the bearing of the line, and is entered into the survey book together with

the horizontal distance of the line and all particulars on each side and along the line.

The second of these methods, fast-needle or *rack* surveying, is adopted when the roadways contain a quantity of iron which may not be removed. The needle is not used at all during the survey, unless at some point where there is no iron a magnetic bearing may be taken for the purpose of reducing the angles to magnetic bearings.

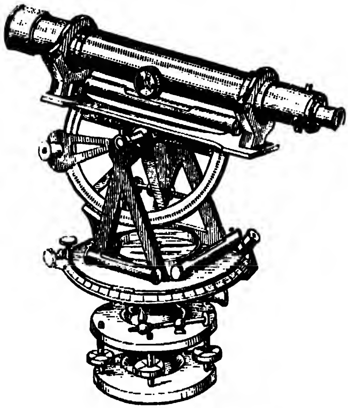


Fig. 230.—Theodolite

In surveying, the angles formed by every two lines are taken and entered in the survey book; the last line is always taken as the base line from which the next angle is measured. Thus the sights of the compass are directed along the first line, and are then turned to look along the second line; the angle formed by the two lines is then read; the compass is then moved to the end of the second line, along which the sights are directed; they are then turned along the third line, and the angle formed by these two lines read, and so on.

There is another method of fast-needle surveying in which all the angular bearings are ascertained from the primary base line, without any connection being made with the magnetic meridian, unless a magnetic bearing is required at some point.

The Theodolite (see fig. 230).—This is the most important and accurate instrument used by the surveyor. It consists essentially of a horizontal circular plate, divided on its outer circumference into degrees and half-degrees.

Upon this plate, and fixed to the vertical axis of the instrument, is another circular plate called the vernier plate, as it carries the verniers. On the vernier plate are two uprights firmly fixed to it carrying the telescope, and a vertical circle which is provided with verniers. The vertical axis of the instrument works freely in a ball and socket between the parallel plates below the horizontal plate. Below the parallel plates is the tripod stand. Upon the vernier plate are two spirit-levels at right angles to each other, and a longer one is attached to the telescope. The vernier plate also carries the compass-box with needle.

Great accuracy is obtained with the theodolite. It is used for surface surveys where exactness is required, as, for instance, boundaries and buildings. It is also used for connecting surface and underground surveys, and for surveying along the principal roadways of mines. The theodolite is not so well adapted for the survey of low workings as the compass, and is only used in them for special purposes.

The Plotting of Surveys.—Plotting is the reproduction on paper, to scale, of the angles and measurements that have been made in surveying either on the surface or underground. The chief instruments used are the protractor, parallel ruler, and scale.

Protractor.—This may be a semicircle of brass with the semi-circumference divided into 180 equal parts, called *degrees*; but it is better when the circle is complete with the circumference divided into 360 degrees. It is generally made with a straight side of brass across the circle to represent the diameter, with a notch in its centre.

Parallel Ruler.—This may be either a crank or a rolling parallel ruler. It is used in plotting for drawing parallel lines, the rolling ruler being preferable.

Scale.—This is an instrument for assisting the work of drawing plans to a reduced scale. Scales of equal parts are made of boxwood or ivory, with a feather edge,

along which they are divided, so that distances can be at once marked off or the length of line ascertained.

The operation of drawing lines on a plan whose length shall be some proportion, say one-half, one-quarter, or any other fraction, of the lines measured on the ground, is called " drawing or plotting to scale ". Mining plans are required to be plotted to the same scale as the large Ordnance Maps of England, viz. 3·15 chains to an inch, or 25 inches to the mile, i.e. every inch on the plan represents a distance of 3·15 chains in the workings in the mine. At many mines the plans are plotted to scales of 2 chains, 3 chains, and 4 chains to an inch, such having been in use at the time of the passing of the Act.

To plot a survey, take a protractor and fix it at any convenient place on the paper, so that its north and south points will coincide with the meridian of the plan. Secure it with a weight, then mark off from the protractor on to the plan the bearings contained in the survey book. Then transfer these bearings from the protractor to where they are wanted on the plan by means of a parallel ruler, measuring off with a proper scale the distances as required. When the survey lines are marked on the plan, the filling in of the offsets and other particulars contained in the survey book may be proceeded with.

The roadways and workings of mines are represented on plans sometimes by single lines only, but more often by double lines—each line of which is the side of a roadway. Engine-planes, wagon-way, and other main roads in coal are shaded, generally with blue colour, and when in stone, by stone colour. The course of faults across the coal-field is frequently denoted in red, and the amount and direction of the throw are stated at the places where the fault is proved. Goaf is generally indicated by cross-hatching.

Levelling.—In order to ascertain the rate of dip of the coal-seam as required by Act of Parliament, the principal roads of the mine are levelled. Levelling is

the art of determining the difference of the heights of two or more points in relation to a level line represented by the level of the instrument. Underground levelling generally commences from the shaft bottom, and the difference of level, above or below, of various points underground in relation to the shaft bottom is ascertained. When the sea-level is known, the difference of level between any of the places underground and the sea-level may also be ascertained.

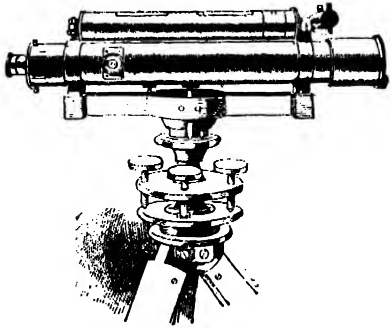


Fig. 231.—Dumpy Level

Levelling is also much used in mines to ascertain the inequalities and undulations of the principal roads, when it is desired to equalize the gradient by taking down some of the roof to fill up the depressions, and by taking up the floor where there are ridges. It is also useful for ascertaining the difference of level of various places in the mine where there is water, when water-levels are to be made or siphons or pumps applied.

An instrument very much used for underground levelling is *Gravatt's*, known as the *Dumpy Level* (fig. 231). It is very simple in construction, and as the diameter of its telescope is considerable and its length short, it gives a good light by which to read the levelling staff, which is a manifest advantage underground.

APPENDIX A

LEGISLATION RELATING TO COAL MINES

It is necessary that all persons employed in mines should make themselves acquainted with the Acts of Parliament and rules in force relating to mines, and especially those rules immediately concerning the duties they are appointed to perform. Every official must be fully acquainted with the rules and regulations respecting his particular office, and the chief officials must have a thorough knowledge of all legislation relating to mining operations, so that they are able not only to observe the laws themselves, but see that all persons under their supervision carry out the various regulations.

It is therefore important that students of mining should make themselves acquainted with all the Acts of Parliament relating to mines.

An Act to Prohibit the Working of Women and Girls in Mines and Collieries, &c., 1842.—This was the first Act of Parliament dealing specially with mines, and had for its principal objects the prevention of the employment of women and girls underground, and the securing of better conditions of work for boys, and the general safety of the mines. It also provided for the appointment of Inspectors of Mines.

An Act to Amend the Law for the Inspection of Coal Mines in Great Britain was passed in 1855.

An Act for the Regulation and Inspection of Mines

was passed in 1860, and was a considerable advancement on its predecessors in its provisions for securing greater safety in the working of mines. Its provisions included the following:

" No boy under twelve years of age to be employed in mines unless he can read and write, or otherwise he must attend school for three hours on two days in every week."

Winding engines, for winding men in shafts, not to be in charge of persons under eighteen years old.

General Rules for the guidance of workers underground were included in this Act, the principal ones referring to ventilation, fencing of dangerous working-places likely to contain gas, examination of safety-lamps, fencing shafts, the lining or casing of shafts to make the sides secure, means for signalling in shafts, refuges to be provided on self-acting and engine planes, cover overhead on cages in shafts, single-link chains prohibited when winding men, flanges to winding drums to prevent slipping, indicator to winding engines, pressure gauges, &c., on boilers, fencing flywheels of engines, boreholes in advance in places approaching dangerous accumulations of water.

Special Rules were provided to be drawn up and established to suit the special requirements of the different mining districts, defining the duties and responsibilities of all officials and workmen.

An Act to Amend the Law relating to Coal Mines came into force on January 1, 1865, making it obligatory to have two shafts or outlets for all mines, such shafts to be not less than 10 feet apart between the sides of each shaft.

Coal Mines Act, 1872.—This was enacted " to consolidate and amend the law relating to the regulation and inspection of coal mines and certain other mines ".

All the chief features of the previous Acts are continued in this Act, but the general rules are much

extended, and a wider range of subjects are dealt with.

The Coal Mines Regulation Act, 1887.—This Act came into force on January 1, 1888, and repealed all former Acts for the regulation of coal mines. It applies to the mines of Great Britain and Ireland, and includes all mines working coal, stratified ironstone, shale, and fireclay.

It is divided into (*a*) General Provisions, (*b*) General Rules, (*c*) Special Rules.

(*a*) *The General Provisions* include many important matters, such as: the hours of employment of boys underground and surface, and of women on surface; the granting of certificates to managers and under-managers; appointment of checkweighers; the provision of more than one shaft; the preparation of plans of shafts and underground workings in relation to surface boundaries; appointment of Inspectors of Mines; holding of inquests and other inquiries into accidents, &c.

The prohibition of single shafts is continued, and the distance between separate shafts is increased to 15 yards instead of 10 feet as in former Acts.

Two classes of managers were created by the provisions of this Act, who must pass examinations for first-class certificates qualifying as manager, or second-class certificates qualifying as under-manager. Every mine must be under the "daily personal supervision" of one or other of these classes.

(*b*) *The General Rules* are thirty-nine in number, and refer to the working of all mines within the meaning of the Act. They appear to cover all matters underground which may be dangerous, with a view to preventing accidents. They deal, amongst other things, with ventilation, making provision for an adequate supply of fresh air; daily inspection of underground workings for gas or other dangers; reports specifying discovery of anything dangerous; use of safety-lamps; conditions on which explosives may be used; approaching old workings likely to contain a dangerous accumulation of water or

gas; timbering working-places and roadways; signals and signalling arrangements in shafts and on roads where haulage is done by machinery; provision of ambulances; barometer and thermometer to be fixed at the entrance of all mines; coal-getters to have had two years' previous employment at the face of the workings.

(*c*) *The Special Rules.*—The Act requires that special rules shall be drawn up and established at each mine for the conduct and guidance of all persons acting in the management of the mine or employed in or about it. As the conditions of mining are not the same in each district, the rules are drafted to suit the special requirements of the various districts, and in general deal with the following:

Certificated manager—his duties and responsibilities in regard to the conduct of the workman and control of the mine; the appointment of officials; the publication of notices, rules, and providing report books, appliances and materials; providing an adequate quantity of ventilation.

Under-manager—his duties and responsibilities under the direction of his superior.

Deputies, firemen, and other officials—their duties and responsibilities in the positions to which they have been appointed by the manager, especially in the examination of the workings for gas and other dangers, and the fixing of timber, firing of shots, &c.

Coal-getters, and all other persons underground—the observance of all rules in regard to the safety of the mine, use of safety-lamps, tools, and appliances, &c.

There are also rules dealing with machinery, haulage roads, timbering, withdrawal of timber, explosives, boilers, ropes, signalling, &c.

The Coal Mines (Checkweighers) Act of 1894.—The checkweigher is appointed and paid by the coal-getters to checkweigh the weighings of the weigher appointed by the mine-owner of all tubs of coal sent out of the mine by the coal-getters. This Act makes it a penal

offence to improperly interfere with the appointment of checkweighers.

The Coal Mines Regulation Act, 1896.—This Act is a short one of the nature of a supplement to the Coal Mines Regulation Act of 1887. The most important provision relates to explosives, and is as follows:

"A Secretary of State, on being satisfied that any explosive is or is likely to become dangerous, may by order, of which notice shall be given in such manner as he may direct, prohibit the use thereof in any mine, or in any class of mines, either absolutely or subject to conditions, and the provisions of the principal Act as to contraventions of General Rules shall apply to contraventions of any such prohibition."

It also contains regulations concerning the use of safety-lamps, the making of plans of mines, the prohibition of coal-dust for stemming explosive charges.

Coal Mines Regulation Act, 1908.—This is "An Act to amend the Coal Mines Regulation Acts, 1887 to 1905, for the purpose of limiting hours of work below ground". It is divided into eight sections, the first being the most important, inasmuch as it limits the hours of work underground to eight, and thus is responsible for this Act being generally known as the "Eight Hours Act".

Section.

1. Limit of hours of work below ground in coal mines. "A workman shall not be below ground in a mine . . . for more than eight hours during any consecutive twenty-four hours."
2. Register of times of descent and ascent.
3. Power to extend hours of work on a limited number of days in a year.
4. Power to suspend Act by Order of Council in event of emergency.
5. Application to mines not entered by a shaft, &c.
6. Provisions for securing compliance with Act.
7. Penalties.

8. Application, commencement, and short title. This Act applies to all mines to which the Coal Mines Regulation Acts, 1887 to 1905, apply, and shall come into operation as respects mines in the counties of Northumberland and Durham on the 1st January, 1910, and elsewhere on the 1st July, 1909.

Coal Industrial Commission Act.—Early in 1919 the miners made certain demands with regard to hours and wages, their policy being to take advantage of a time of national stress to precipitate a crisis and compel the Government to nationalize the mines. A Royal Commission was formed under Mr. Justice Sankey, and after one of the most farcical inquiries that ever took place in this country the Miners were given an advance of wages, and their hours reduced from 8 to 7, with the promise of a further reduction to 6 per day in 1921. The Eight Hours Act was thus converted into a Seven Hours Act, and may yet become a Six Hours Act.

Mine Accidents (Rescue and Aid) Act, 1910.—This is an Act to make provision with respect to organization for the purpose of rescue and aid in the case of accidents in mines.

The Secretary of State may, by order, of which notice shall be given in such manner as he may direct, require such provision as he may consider to be necessary to be made at all mines or any class of mines in regard to all or any of the following matters:

(*a*) Supply and maintenance of appliances for use in rescue work, and formation and training of rescue brigades.

(*b*) Supply and maintenance of ambulance appliances and the training of men in ambulance work.

Special Rules for the Installation and Use of Electricity in Mines.—These came into force in 1912. The rules are numerous, and are divided into twenty sections in addition to " Duties " and " Definitions ".

1. The Use of Electricity prohibited in Certain Places. The use of electricity is prohibited in any part of a mine where, on account of the risk of explosion of gas or coal-dust, such use would be dangerous to life.
2. Notices for H.M. Inspector. These refer to the introduction of electricity into any mine, and an annual return is required of size and type of apparatus in use underground.
3. Plan. This has to show the position of all fixed apparatus in the mine.
4. Notices for Workmen. These refer to handling or interfering with apparatus, and directions as to the restoration of persons suffering from the effects of electric shock.
5. Lighting, Telephones, and Fire Buckets. These rules refer to shaft bottoms, telephonic communications between surface and shaft bottom, and the keeping of dry sand in case of fire.
6. Housing of Apparatus and Working Space. These have reference to the construction of motor houses and material used.
7. The Construction of Apparatus and the Insulation of a System. These are important, having reference to the manufacture of apparatus, which must be so constructed, protected, worked, and maintained as to prevent danger. Also, must be kept efficiently insulated from earth, and means provided for indicating any defects in the insulation of a system.
8. Earthing. These refer to the earthing of all apparatus underground to an earthing system at the surface of the mine.
9. Use of High or Extra-high Pressure Current.
10. Switchgear. These have reference to the construction and fixing where there may be risk of igniting gas, coal-dust, or other inflammable material.

11. Control of the Supply of Current. These are important, dealing with means for cutting off all pressure, the stopping and starting of motors, lightning arrestors, &c.
12. Cables. These deal with the construction of the insulating material, the protection of the cables from mechanical damage, the joining up to motors, transformers, switchgear, and other apparatus.
13. Portable Apparatus.
14. Supervision and Working of Apparatus. These are important, having reference to the daily examination, testing, and adjusting by competent electricians of all electric apparatus underground. The electrician shall keep a daily logbook. These rules also refer to the keeping of all apparatus clear of obstruction and free from dust, and the avoidance of handling apparatus electrically charged.
15. The Use of Electricity where Inflammable Gas is likely to be present. In any part of a mine in which inflammable gas, although not normally present, is likely to occur in quantity sufficient to be indicative of danger, the following shall be observed:

    All cables, apparatus, signalling wires, and signalling instruments shall be constructed, installed, protected, worked, and maintained so that in the normal working thereof there shall be no risk of open sparking.

    Every electric lamp shall be enclosed in an air-tight fitting, and the lamp globe itself shall be hermetically sealed. A safety-lamp shall be provided and used with each motor when working, and should any indication of firedamp appear from such safety-lamp, the person appointed to work the motor shall forthwith cut off the pressure therefrom and report the matter to an official.

16. Shot-firing.
17. Signalling. Where electricity is used for signalling, the pressure in any one circuit shall not exceed 25 volts.
18. Electric Relighting of Safety-lamps. These rules refer to the relighting apparatus wherever in use underground for the relighting of safety-lamps.
19. Locomotives. Overhead trolley-wire system haulage is prohibited in any part of a coal-mine.
20. Exemptions. These relate to certain apparatus which had been in use prior to 1st June, 1911.

Coal Mines Act, 1911.—An Act to consolidate and amend the Law relating to Coal Mines and certain other mines. This Act, passed on 16th December, 1911, came into operation on the 1st July, 1912. It applies to mines of coal, mines of stratified ironstone, mines of shale, and mines of fireclay. It is a great advance upon any previous mining legislation. In addition to all the regulations contained in the Act, the Secretary of State has power to issue " orders " containing further regulations or rules for the prevention of accidents. " Orders " have been issued such as the " Explosives Order " and " Rescue Order ". Also, he has power to issue special regulations which may supplement or modify the general regulations which provide for the safety, health, convenience, and proper discipline of the persons employed in or about mines, and for the care and treatment of horses and other animals used therein, and altogether, in a word, gives the Mining Industry Government by Order instead of by Act of Parliament.

The Act is divided into eight parts.

Part I deals with the management, and the qualifications of managers, under-managers, firemen, examiners and deputies, inspections on behalf of workmen, and the returns, plans, notices, and books required.

Part II deals with the provisions as to safety under-

ground. The most important subjects are: Ventilation, Safety-lamps, Shafts and Winding, Travelling Roads and Haulage, and Support of Roof and Sides. There are provisions as to Machinery, Explosives, and Prevention of Coal-dust. It also comprises regulations as to Inspections of Working-places, Examinations of Machinery, Boring to Old Workings, and provision of means for extinguishing fire.

The provisions included in this part practically cover the general working arrangements and management of mines underground, and the means of safeguarding life and property.

Part III deals with the arrangements affecting the health of the workmen. The chief provisions refer to sanitary conveniences, washing and drying accommodation, use of sprays when drilling siliceous rock to prevent the escape of dust into the air if the dust is liable to give rise to fibroid phthisis, and notification of specified industrial diseases.

Part IV. This refers to notices required to be given of any accidents to persons, or certain dangerous occurrences, and the reports, investigations, and inquests that may be necessary according to the nature and results of the accidents.

Part V contains provisions as to the making of General and Special Regulations, and the publication of abstract of Act and Special Rules, and penalties for breaches of regulations.

Part VI deals with the employment of boys, girls, and women, such as: the prohibition against employment of boys, girls, and women underground, and the conditions under which they may be employed at the surface of mines.

Under this part are also included the provisions as to the weekly payment of wages.

Part VII defines the powers and duties of Inspectors of Mines.

Part VIII. Supplemental. Legal proceedings for offences against the Act either by owners or their agents and managers, or workmen.

Miscellaneous. There are twenty-eight sections under this heading, and these include such important matters as the care and treatment of horses and other animals in mines; powers of Secretary of State as to making and revoking orders and exemptions; procedure for ascertaining and certifying the views of workmen; and definition of terms used in the Act.

There are four Schedules to the Act. The first contains the forms required to be sent in respect of each colliery to the Inspector of the District on or before the 21st January every year. The second contains the procedure for making General Regulations. The third contains the rules for the care, treatment, and examination of horses employed underground. There are seventeen rules under this schedule. The fourth schedule contains the enactments repealed by this Act. They are as follows:

The Coal Mines Regulation Act, 1887. Repealed, except sections 1, 3, 12, 13, 14, and 15.

The Coal Mines Regulation Act, 1896. The whole Act repealed.

The Mines (Prohibition of Child Labour Underground) Act, 1900. The whole Act repealed.

The Coal Mines Regulation Act (1887) Amendment Act, 1903. The whole Act repealed.

The Notice of Accidents Act, 1906. Sections 1, 2, 3, and 5 repealed.

The Employment of Women Act, 1907. Section 1 repealed.

The Mines Accidents (Rescue and Aid) Act, 1910. The whole Act repealed.

Mining Industry Act, 1920.—This is an Act for securing the most effective development and utilization of the mineral resources of the United Kingdom, and the safety

and welfare of those engaged in the mining industry. It is divided into three parts.

Part I deals with administration and establishes a new Mines Department under the Board of Trade; it provides powers for regulating the export price of coal, for the formation of Advisory Committees, for the transfer of officials presently engaged under the Coal Control to the Board of Trade with their present salaries and pensions, and sets aside a sum of £250,000 for the payment of further officers and servants. It also gives powers to the holder of the office of Secretary for Mines to sit in Parliament if he can find a constituency willing to elect him.

Part II makes provision for the establishment of Pit and District Committees and Area and National Boards, on the former two of which one-half of the representation is by workmen, who are to be paid fees for their services by the owner of the mine. When recommendations made by an Area Board or by the National Board are accepted by the Board of Trade, then the Board of Trade has powers to compel any person to comply with such directions as they may issue to give effect to those recommendations.

Part III makes provision for carrying out any schemes as to drainage of groups of mines. It also establishes a fund to which the owner of the mine must contribute one penny per ton on the coal raised by him for such purposes connected with the social well-being, recreations, and conditions of living of workers in or about coal mines, and with mining education and research, as the Board of Trade, after consultation with any Government department concerned, may approve. It further gives the Board of Trade powers to hold inquiries, and to compel any person to give evidence or to produce documents which may relate to such inquiries if he has such in his possession.

Altogether the Act establishes a new bureaucracy, and

the future will show whether the officials to be employed under it, or the coal trade, will benefit most.

Truck Acts, 1831, 1887, 1896.—The first Truck Act was passed to prohibit payment to a workman in kind instead of current coin, and without deductions except those to which he has agreed. Deductions may be made for materials, tools, and machines supplied to the workmen, or for foreign matter sent out of the mine other than the mineral contracted to be gotten. The later Acts particularize the terms which must be made with workmen in reference to deductions from wages.

The Workmen's Compensation Act, 1906.—An Act to consolidate and amend the law with respect to compensation to workmen for injuries suffered in the course of their employment.

The principal provisions are:

1. Liability of employers to workmen for injuries. An employer is liable under this Act to pay compensation if the workman sustains an injury at work which disabies him for a full week or more. If it is proved that the injury to a workman is attributable to the serious and wilful misconduct of that workman, any compensation claimed in respect of that injury shall, unless the injury results in death or serious and permanent disablement, be disallowed.
2. Time for taking proceedings.
3. Contracting out.
4. Sub-contracting.
5. Provision as to cases of bankruptcy of employer.
6. Remedies both against employer and stranger.
7. Application of Act.
8. Application of Act to industrial diseases.
9. Application of Act in employment of Crown.
10. Appointment and remuneration of medical referees and arbitrators.
11. Returns as to compensation.
12. Definitions.

13. Special provisions as to Scotland.
14. Provisions as to existing contracts and schemes.

There are three schedules to the Act.

The first schedule deals with the scale and conditions of compensation; the second, arbitration; and the third, with a description of diseases for which compensation is payable, and of the industrial processes which may give rise to the diseases enumerated.

There are several other Acts which contain provisions affecting the working and management of mines, chief of which are the following:

The Explosives Act, 1875.—"An Act to amend the law with respect to manufacturing, keeping, selling, carrying, and importing gunpowder, nitro-glycerine, and other explosive substances."

The Factory and Workshop Act, 1901 (Non-textile Factories).

The Boiler Explosions Act, 1882 (Amended 1890).

The Petroleum Act, 1871.

The Alkali, &c., Works, Regulation Act, 1906.—Works where gas-tar is distilled or is heated in any manufacturing process, or gas liquor and sulphate of ammonia works, come under the provisions of this Act. As by-product recovery works at coke ovens are in operation in connection with many collieries, they come within the sphere of this Act.

APPENDIX B

FIREMEN'S, EXAMINERS', AND DEPUTIES' CERTIFICATES

A full certificate for a fireman, examiner, or deputy certifies three things:

1. That he can test for gas with a safety-lamp.
2. That he can measure an air current.
3. That his hearing is good.

All firemen, &c., appointed *after the passing of the Act* (i.e. after 16th December, 1911) have to obtain the full certificate, with the exception only of firemen, &c., employed in mines in which inflammable gas is unknown, who are required to obtain a certificate as to 2 and 3 only, and not as to 1.

Gas Tests.—A candidate will be required to show that he is able to make accurate tests for inflammable gas, and that he can do this with any of the various kinds of safety-lamp, and with any of the various kinds of oil, *in use in the district in which he is employed*. For this purpose he will be required, first, to show to the examiner that he understands the method of detecting inflammable gas by means of a safety-lamp; and, secondly, to make correct observations of gas caps as they appear on the flames of various kinds of safety-lamps burning the various kinds of oil. By a correct observation is meant that the candidate must be able to judge by the size and appearance of the cap how much gas approximately is present. He will be tested with percentages of gas from 2 per cent upwards, and no candidate will pass who is unable to see a 2 per cent cap. (See Chapter XXIV.)

A candidate who ordinarily wears glasses will not necessarily be called upon to make the observations without them, but it will rest with the examiner to decide by such test as he thinks fit whether the eyesight is so defective that under the working conditions in a mine a candidate would not be able to make accurate observations.

Air Measurements.—A candidate will be required to show that he can measure the quantity of air in an air current. The quantity of air, in cubic feet per minute, passing through the air-way of a mine being found by multiplying the number of square feet in the cross-section of the air-way by the speed of the air current in feet per minute, the candidate will be required, first, to measure the speed of the air current passing through the air-way of a mine, or through some gallery corresponding to or representing the air-way of a mine, both (i) by means of an anemometer, and (ii) by observing the speed at which dust or smoke is carried by the air current; and, secondly, to calculate from measurements the size (i.e. cross-sectional area) of the air-way or gallery in which the air to be measured is passing. (See Chapter XXIII.)

Hearing.—The candidate's hearing will be tested. The Act requires that his hearing should be such as to enable him to carry out his duties efficiently. This does not mean that a fireman must possess a standard of hearing above that of the average man, nor is it required that his hearing must be up to the standard in both ears. If the candidate is able to hear sounds audible to the average man it will be sufficient. The tests will usually be of a simple nature, and will naturally be varied by the examiner with different candidates and at different times.

A candidate who claims to be exempted from Part I of the examination must produce to the school or examining centre to which he presents himself for

examination a certificate from the manager of his mine that he is employed in a mine in which firedamp is unknown. Note, however, that a fireman, &c., employed in a mine in which gas is unknown, who claims exemption from Part I, would have to pass this part of the examination afterwards, if at any time he obtained employment as a fireman, &c., in a gassy mine.

The Act provides also for further certificates as to eyesight and hearing to be obtained by a fireman, examiner, or deputy at intervals of five years. The object of these further certificates is to ensure that a fireman's eyesight continues to be such as to enable him to make accurate tests for firedamp, and that his hearing continues to be such as to enable him to carry out his duties efficiently. The first of these certificates will not be required until the expiration of five years from the date on which he got his fireman's certificate as above. A fireman, examiner, or deputy will be able to get this five-yearly certificate either from one of the schools, institutions, or authorities approved by the Secretary of State, or from a doctor selected by himself. The fee for obtaining this certificate will be payable by the fireman's employer.

EXAMPLES WITH ANSWERS

1. Find the approximate temperature of the strata in degrees Fahrenheit at a depth of 3500 feet from the surface. *Ans.* 107° F.

2. At what depth from the surface will the underground temperature of the strata register 80° F.? *Ans.* 1850 feet.

3. A pitch-pine frame dam of 15 feet radius is used to hold back water at a pressure of 260 lb. per square inch. Taking the safe compressive strength of pitch pine as 650 lb. per square inch, find the thickness of the dam. *Ans.* 99 inches, or 8 feet 3 inches.

4. An ordinary brick dam of 12 feet radius is used to hold back water under a pressure of 32 lb. per square inch. If the safe compressive strength of the brick is 80 lb. per square inch, find the required thickness for the dam. *Ans.* 89 inches.

5. Find the cost of a bore-hole 240 fathoms deep if the rate for the first 5 fathoms is 10*s.* per fathom, with an increase of 5*s.* per fathom for each additional 5 fathoms. *Ans.* £1530.

6. Find the total cost of a bore-hole 160 fathoms deep if the rate for the first 5 fathoms is 7*s.* 6*d.* per fathom, with an increase of 7*s.* 6*d.* per fathom for each additional 5 fathoms. *Ans.* £990.

7. A bore-hole costs 13*s.* 4*d.* per foot for the first hundred feet, 16*s.* 8*d.* per foot for the second hundred feet, 20*s.* per foot for the third hundred feet, and so on. If the total cost is £1900, find the depth. *Ans.* 1200 feet.

8. Find the pressure in pounds per square inch due to a head of 108 fathoms of water. *Ans.* 281·25 lb.

9. At what depth will the pressure due to a vertical column of water amount to 250 lb. per square inch? *Ans.* 96 fathoms.

10. Find, using Greenwell's formula, the thickness of tubbing required to withstand a pressure due to 300 feet head of water in a shaft 20 feet internal diameter. *Ans.* 1·8 inches.

11. Find the thickness of tubbing required to stand the

pressure of 100 fathoms head of water in a shaft 15 feet diameter. *Ans.* 2·52 inches.

12. A shaft 18 feet diameter is lined with cast-iron tubbing 2·952 inches thick; what head of water will this tubbing stand? *Ans.* 600 feet.

13. Find the centigrade and Réaumur readings corresponding to 60° F. *Ans.* 15·55° C. and 12·4° R.

14. The temperature of the air in the intake air-way of a certain mine is measured as 18° C.; what is the temperature expressed in degrees Fahrenheit? *Ans.* 64·4° F.

15. The freezing-point of mercury is readily remembered because it is given by the same numbers on the Fahrenheit and centigrade scale. Find this temperature. *Ans.* − 40°.

16. Calculate the temperatures in degrees centigrade corresponding to 100° F. and 0° F. *Ans.* 37·7° C. and − 17·7° C.

17. How many foot-pounds of work must be expended in pumping 40 cubic feet of water weighing 62·5 lb. per cubic foot to a height of 35 feet? *Ans.* 87,500 foot-pounds.

18. Find the work done in winding a chain 250 feet long and weighing 10 lb. per foot up on to a drum. *Ans.* 312,500 foot-pounds.

19. If 500 cubic feet of water is raised per minute to a height of 200 feet, what is the work done per minute in foot-pounds, and at what rate in horse-power is it being done? *Ans.* 6,250,000 foot-pounds and 189·39 h.p.

20. A colliery engine raises a loaded cage from a depth of 200 fathoms in 45 seconds. If the unbalanced load on the engine is 21 cwt., find the horse-power required. *Ans.* 114 h.p.

21. What horse-power is required to lift 3000 cubic feet of water per hour to a height of 160 feet, supposing one-quarter of the power to be lost by the friction of the pumps? *Ans.* 20·2 h.p.

22. A drum 6 feet diameter is used for raising a load of 2240 lb. After how many revolutions will 565·48 foot-tons of work have been expended? *Ans.* 30 revolutions.

23. The piston of a haulage engine is 2 feet diameter and the stroke 3 feet. If the pressure of steam is 160 lb. per square inch, find the work done in 1000 strokes. *Ans.* 96,979 foot-tons.

24. A winding engine indicates 90 h.p. when it is raising a load of 3 tons at the rate of 300 feet per minute. Find the fraction of the power lost by friction. *Ans.* 0·32.

25. An engine has a cylinder 10 inches diameter and a stroke of 2 feet. If the mean steam pressure is 45 lb. per square inch, and the engine makes 90 revolutions per minute, find its indicated horse-power. *Ans.* 38·5 i.h.p.

26. An engine has to develop 120 i.h.p. If the steam be 60 lb. per square inch, the length of the stroke 3 feet, and the

number of revolutions 55 per minute, find diameter of cylinder. *Ans.* 16 inches.

27. An engine has a piston 12 inches diameter and a stroke of 18 inches. Find the number of strokes that it must make per minute when developing 20 i.h.p., the mean steam pressure being 22 lb. per square inch. *Ans.* 177 strokes.

28. What would be the indicated horse-power of a high-pressure non-condensing steam-engine in which the mean effective pressure of the steam is 25 lb. per square inch, the diameter of the cylinder 10 inches, and the length of the stroke 18 inches when running at 120 revolutions per minute? *Ans.* 32 i.h.p.

29. Calculate the weight of a cubic foot of dry air at a temperature of 60° F. and a pressure of 30 inches barometer. *Ans.* ·0766 lb.

30. Find the weight of a cubic foot of air at a temperature of 12° C. and a barometric pressure of 29·38 inches. *Ans.* ·0778 lb.

31. The temperature of a downcast shaft is 49° F., while that of the upcast is 95° F. If the barometer reads 29·88 inches, find the difference in weight of air per cubic foot in the two shafts. *Ans.* ·00647 lb.

32. Find the pressure of the atmosphere in pounds per square inch when the barometer registers 29·8 inches. *Ans.* 14·63 lb. per square inch.

33. The reading of the barometer at the top of a shaft is 30 inches, while at the bottom the barometer reads 32 inches. Find the depth of the shaft. *Ans.* 583·8 yards.

34. A barometer reads 30 inches at the top of a shaft which is 400 yards deep. What will it read at the bottom, assuming the same temperature in both cases? *Ans.* 31·37 inches.

35. A dook dips 1 in 6 along a line bearing due south. What will be the bearing of a road set off from the dook to rise 1 in 50 in an easterly direction? *Ans.* N. 83° 7′ W.

36. Prove that the height of the water-gauge in inches multiplied by 5·206 = the ventilating pressure in pounds per square foot.

37. An air-way is 10 feet by 6 feet, and 500 fathoms long. Find the pressure required to pass 30,000 cubic feet of air per minute; take $k = \cdot 0217$. *Ans.* 8·68 lb.

38. Find the quantity of air passing through an air-way 6 feet 6 inches high by 6 feet 3 inches wide, when the velocity is 640 feet per minute. If the velocity falls to 560 feet per minute, find the quantity that will then pass. *Ans.* 26,000 cubic feet per minute and 22,750 cubic feet per minute.

39. Find the horse-power required to pass 30,000 cubic feet of air per minute through an air-way when the ventilating pressure is 5 lb. per square foot. *Ans.* 4·54 h.p.

40. If instead of 30,000 cubic feet per minute as in the last example, we desire to pass 60,000 cubic feet per minute, what pressure and horse-power will now be required? *Ans.* 20 lb. per square foot and 36·36 h.p.

41. If 20,000 cubic feet of air per minute is passed in a certain air-way by a certain pressure and power, what will be the ratios of the pressures and powers to pass (*a*) 9000 cubic feet of air per minute, (*b*) 40,000 cubic feet of air per minute in the same air-way? *Ans.* (*a*) ·2025 of the original pressure and ·0911 of the original horse-power; (*b*) 4 times the original pressure and 8 times the original horse-power.

42. An air-way is 12 feet by 6 feet, and 1000 fathoms long. What pressure, water-gauge, and horse-power will be required to pass 10,000 cubic feet of air per minute through it? *Ans.* 1·255 lb. per square foot, 0·2415 inches water-gauge, and 0·324 h.p.

43. A colliery fan circulates 120,000 cubic feet of air per minute at a water-gauge of 3 inches. Find (*a*) the horse-power in the air, and (*b*) the horse-power of the fan if its efficiency is 60 per cent. *Ans.* (*a*) 56·7 h.p. in the air; (*b*) 94·5 h.p. to drive fan.

44. A fan circulates 165,000 cubic feet of air per minute at 3 inches water-gauge. If it requires 130 b.h.p. to drive it, what is its efficiency? *Ans.* 60 per cent.

45. How much more resistance does an air current encounter when travelling at 500 feet per minute than at 400 feet per minute? If the water-gauge is 1 inch at the lower velocity, how much would it be at the larger? *Ans.* 1·56 times and 1·56 inches.

46. The water-gauge at the fan drift of a colliery is 2·5 inches, and a current of 140,000 cubic feet of air per minute is passing. If the combined efficiency of the fan and the driving engine is 52 per cent, what is the indicated horse-power of the engine? *Ans.* 106 i.h.p.

47. A fan runs at 250 revolutions per minute, and produces 45,000 cubic feet of air per minute with a water-gauge of 5 inches. Find the horse-power in the air. If this quantity is increased to 60,000 cubic feet per minute without altering the air-ways, find (*a*) the water-gauge, and (*b*) the horse-power. *Ans.* 35·45 h.p.; (*a*) 8·88 inches water-gauge, and (*b*) 84 h.p.

48. A Guibal fan 35 feet diameter runs at 45 revolutions per minute, and produces 2 inches water-gauge. Find its efficiency. *Ans.* 65·36 per cent.

49. If 30,000 cubic feet of air passes per minute through an air-way 8 feet wide by 5·5 feet high and 3000 feet long, what quantity would pass through a similar length of air-way whose width is 10 feet and height 7 feet, with the same pressure? *Ans.* 53,600 cubic feet per minute.

50. A rectangular air-way 10 feet wide and 6 feet high has an air current passing at a velocity of 120 feet per minute. Find the quantity in cubic feet per minute. *Ans.* 10,800 cubic feet per minute.

51. An air-way 10 feet high and 10 feet wide has a distance of 50 feet measured off along a portion of its length where the section is uniform. A small piece of fuse is fired at the inlet end, and it is noted that it takes the smoke 20 seconds to travel over the 50 feet measured off. Find the quantity of air passing. *Ans.* 15,000 cubic feet per minute.

52. It is noted that smoke travels 80 feet in 25 seconds in a road the section of which is 8·5 feet by 6·5 feet. Find the quantity of air passing in cubic feet per minute. *Ans.* 10,608 cubic feet per minute.

53. An anemometer reads 3125 feet before being set in motion; after running for 1 minute in an air-way 8 feet by 6·5 feet, it reads 3250 feet. Find the quantity of air passing. *Ans.* 6500 cubic feet per minute.

54. An air-way has an area of 72 sq. feet. An anemometer reads 350 feet before being set free, and finally at the end of a 3-minute run it reads 1100. Find the quantity of air passing per minute. *Ans.* 18,000 cubic feet per minute.

55. The following five complete readings of one minute duration each were taken in a roadway the height of which was 6 feet and the width 9 feet. Find the quantity of air passing in cubic feet per minute. (1) First reading 172; second reading 280. (2) First reading 280; second reading 391. (3) First reading 391; second reading 505. (4) First reading 505; second reading 607. (5) First reading 607; second reading 717. *Ans.* 5886 cubic feet per minute.

56. Find the best inclination for a horse-road laid with tram rails (friction 60 lb. per ton) when tub weighs 4 cwt. and contains 10 cwt. of coal. *Ans.* 1 in 67.

57. Find the best inclination for a horse-road when laid with bridge rails (friction 32 lb. per ton) when tub weighs 4 cwt. and contains 10 cwt. of coal. *Ans.* 1 in 126.

58. Find the flattest inclination at which an incline will self-act when the rake consists of 12 tubs each weighing 4 cwt. and holding 10 cwt. of coal. The road is laid with bridge rails ($f = 32$ lb. per ton), and is 300 fathoms long, the rope used being $2\frac{1}{2}$ lb. per fathom. *Ans.* Inclination must be steeper than 1 in 33·5.

59. Find the horse-power required to haul 350 tons of coal per day of 7 hours by direct haulage up an incline 500 fathoms long, rising 1 in 9. Rope 10 lb. per fathom, friction 32 lb. per ton, rope friction $\frac{1}{10}$ of rope weight. Tubs 4 cwt. empty, 14 cwt. gross. *Ans.* About 120 h.p. if speed is taken at 8 miles per hour.

60. An output of 200 tons of coal has to be hauled from a dook 500 yards long, with an inclination of 1 in 10, in $7\frac{1}{2}$ hours. Tubs hold 10 cwt. of coal and weigh 4 cwt. empty. Rope weighs 4 lb. per fathom. Speed 10 miles per hour. Find the horse-power required. *Ans.* 52 h.p. at haulage rope.

61. Find the horse-power required to draw a rake of 6 hutches up a gradient of 1 in 5 at a speed of 10 miles per hour, if the road is 500 fathoms long and laid with bridge rails. The rope weighs 6 lb. per fathom, and the tubs weigh 14 cwt. gross and have a tare of 4 cwt. *Ans.* 74 h.p.

62. Find the size and weight per fathom of a plough-steel wire rope to lift 5 tons of a dead load, the factor of safety being 10. *Ans.* Rope 3·5 inches circumference, weight 12·5 lbs. per fathom.

63. The following is the cage load on a winding drum the smaller diameter of which is fixed as 12 feet. Find the larger diameter in order to obtain a balanced wind. Cage 1 ton, coal 30 cwt., tubs 12 cwt., rope 28 cwt. *Ans.* Large diameter = $19\frac{7}{17}$ feet.

64. Find the size of bucket-pump required to raise 200 gallons of water per minute when the slip is $12\frac{1}{2}$ per cent, and the pumping speed 30 feet per minute. *Ans.* 15 inches diameter.

65. Find the total pressure on the ram of a 9-inch pump when forcing water through a column of pipes 200 fathoms vertical. *Ans.* 33,048 lb.

66. What size of pump is required to raise 260 gallons of water per minute, if the slip be 10 per cent and the pumping speed 30 feet per minute? *Ans.* 17 inches diameter.

67. What is the diameter of a pump with a useful effect of 85 per cent when working at 15 effective strokes per minute, the length of the stroke being 2 feet, and the number of gallons of water to be dealt with per minute 125? *Ans.* 12 inches diameter.

68. Find the size of each of the rams of a 3-throw ram pump to raise 240 gallons of water per minute, if the pumping speed be 60 feet per minute and the slip 15 per cent. *Ans.* Each ram approximately 7 inches diameter.

69. Find the horse-power required to raise 240 gallons of water per minute to a height of 300 feet vertical. *Ans.* 21·8 h.p.

70. A single acting pump is making 10 effective strokes of $4\frac{1}{2}$ feet each minute. If the ram has a diameter of 15 inches, how many gallons of water will it raise per minute? *Ans.* 345 gallons.

71. It is desired to send a current of 80 amperes through a cable the total resistance of which is 5 ohms. Find the voltage required. *Ans.* 400 volts.

72. If a current of 75 amperes is passing in a circuit where

the pressure is 450 volts, what is the resistance of the circuit? *Ans.* 6 ohms.

73. Find the volts lost in $\frac{1}{2}$ mile of $\frac{7}{14}$ cable the resistance of which is 1·214 ohms per mile. *Ans.* 21·25 volts.

74. Find the watts lost when a current of 40 amperes is passed through a resistance of 2 ohms. *Ans.* 3200 watts.

75. The current through a carbon filament lamp is 0·65 ampere and the voltage 130. What is the resistance of the lamp? *Ans.* 200 ohms.

76. A certain dynamo generates 270 amperes at 500 volts. Express this output (*a*) in horse-power, (*b*) in kilowatts. *Ans.* (*a*) 181 h.p. nearly, (*b*) 135 kilowatts.

77. A dynamo generates 540 amperes at 500 volts. If the combined efficiency of the driving engine and dynamo is 80 per cent, find the indicated horse-power of the driving engine. *Ans.* 452 i.h.p.

78. A centrifugal pump has to raise 500 gallons of water per minute to a height of 50 feet. Find the brake horse-power of the motor required if the efficiency of the pump be 70 per cent. *Ans.* 11 b.h.p.

79. Find the brake horse-power of motor required to drive a pump which raises 240 gallons of water per minute against a head of 500 feet if the efficiency of the pump is 90 per cent. *Ans.* 40·4 b.h.p.

80. If 55 electric miners' lamps are to have their batteries charged from a 200-volt circuit, and the charging current is not to exceed 1·5 amperes, find the resistance that must be put in circuit with the batteries if each battery has an e.m.f. of 2·6 volts, and has an internal resistance of 0·04 of an ohm. *Ans.* A resistance of 35·8 ohms.

Note that in a considerable number of the foregoing questions, the answers are only approximately correct, so that students need not worry although their results may differ by a decimal point or two.

GLOSSARY

Adit. A water-level drift.

After-damp. The gaseous mixture resulting from an explosion of firedamp.

Air Compressor. An engine to compress air and increase its pressure above that of the atmosphere.

Air-ways. The roads along which currents of air travel in mines.

Anthracite. Coal high in carbon with a small percentage of volatile matter.

Aqueous Rocks. Those which have been formed by the agency of water.

Atmospheric Pressure. The weight of the air at the surface of the earth.

Back. A plane or cleavage in the coal.

Backing Deals. Deals of wood placed perpendicularly behind cribs in a shaft.

Balk. A thick piece of timber fixed under the roof of a road to prevent the stone falling.

Banking out. The operation of changing the tubs in the cages at the surface.

Banksman. The man in charge of signals at the top of shaft, and who changes the tubs when the cages reach the surface.

Barrier. A rib or tract of coal left unworked to separate one district from another or one colliery from another.

Basin. When beds dip from all directions towards the centre of a hollow, they form a geological basin.

Bit. The working edge or steel tip of a drill or borer.

Black Damp. A mixture of carbonic acid gas and nitrogen.

Blasting, or **Shooting.** The operation of breaking away masses of coal or stone by means of an explosive.

Blower. A sudden outburst of a flow of firedamp.

Bord. An excavation driven across the line of the cleat of the coal-seam.

Bore-hole. A hole of small diameter bored or drilled in coal or stone horizontally, vertically, or obliquely.

Borer. The tool used for boring holes for blasting in mines.

Brace Head. Wooden handles for raising and turning boring rods.

Brattice. A partition made in a shaft, working-place, drift, or level to separate the excavation into two equal or unequal compartments, one of which acts as an intake and the other as a return.

Broken Working. The working away or removal of blocks or pillars of coal formed by whole workings.

Bucket. The valved piston attached to the spears of a lifting sett.

Buntons. Beams of wood placed horizontally across a shaft and secured in the sides.

Carboniferous. Coal or carbon yielding. More particularly applied to that formation of Palæozoic rocks which contain a very large quantity of coal.

Chocks. Pieces of hard wood built in a square between roof and thill to prevent a fall of stone.

Clack. A valve used in pumps.

Cleats. Pieces of wood nailed across the ends of two segments of cribs to secure them together.

Coal Royalties. The areas of coal leased to the proprietors of the mine by the owners of the minerals below the surface.

Core. Cylindrical pieces of rock produced in diamond drilling.

Core Drill. Any type of drill that produces cores of the strata drilled through.

Cribs. Segments of oak to encircle the shaft.

Crosscut. An excavation driven at an angle between a bord and a headways.

Cross-gateways. Roads cutting across ordinary gateways.

Crowntrees. Half-round pieces of timber fixed under the roof of ordinary working-places.

Curb. The temporary support used in a circular shaft.

Delivery. The point where the stream of water issues from a pump or pipe.

Denudation. The wearing down of rocks and the removal of the matter in mechanical suspension or chemical solution.

Deposit. An accumulation of rock matter which has settled down from suspension in water.

Dip, or Fall. The inclination of the rock beds and seams of coal.

District. A number of working-places near together in a separate portion of the area of coal to be worked.

Door-piece. The part of a pump which provides access to the valves.

Downcast Shaft. The shaft down which fresh air passes into the workings of the mine. This is usually made the winding shaft.

Drift. Any excavation in coal or stone underground, which is being driven, or in course of extension, for any purpose other than that of actual coal-getting, is frequently termed a drift. Thus there are coal drifts and stone drifts, either horizontal or inclined.

Drill. An instrument used in boring holes.

Drum. The barrel upon which a rope is wound.

Endless Rope. A system of haulage used in mines.

Engineman, or **Brakesman.** The man in charge of winding engine.

Engine-plane. A road in the mine along which tubs are conveyed by machinery.

Exploring Drift, or **Place.** The working drift approaching old workings whose exact position is uncertain, bored as a precaution against an unexpected holing.

Face. The extreme end of any excavation in the mine at which the men work in extending it, as, for instance, in the operation of getting coals. Thus we speak of the face of a drift or other working-place. The " face " is sometimes termed the " end ".

Fan. A machine placed at the top of a shaft to exhaust the air from the mine and so produce a circulation.

Feeders. Runners of water issuing from the strata in mines. *Surface* feeders are those which have direct communication with the surface, and vary in quantity with the conditions of the weather. *Partial* feeders are in communication with cavities containing water, which decrease or become exhausted. *Permanent* feeders continue without diminution, and have an inexhaustible supply.

Fillers. The men employed in filling into tubs coal which has been broken down or detached.

Fireman. The official who examines the mine to detect the presence of inflammable gas.

Flitting. Removing a coal-cutting machine across gateways from one part of the face of a longwall to another.

Floor, or **Thill,** or **Bottom.** The bed of stone upon which the coal-seam rests. It usually consists of fire-clay.

Fore Drift. Drifts are usually arranged in pairs, and the one which leads or is kept in advance of the other is termed the fore drift and the latter the back drift. The fore drift is usually the intake and the back drift the return.

Formation. A series of strata which are more or less alike as regards the kind of rock and fossils.

Fossil. The petrified remains, or impressions of the remains, of animals and plants found embedded in rocks.

Furnace. A fire placed at the bottom of a shaft to rarefy the air and so produce ventilation.

Fuse. A waterproof cord containing a train of gunpowder for firing explosives in blasting.

Gad. A wedge used in breaking down coal.

Gas. Carburetted hydrogen or firedamp is generally termed simply " gas " in mining districts.

Gateways. Roads maintained through the goaf of a seam worked by the longwall method.

Getters. The men employed in breaking down or detaching coal after it has been holed.

Goaf, Gob, or **Waste.** The area from which the coal has been entirely extracted, and in which the roof has been allowed to fall.

Grit. Usually a hard sandstone in which the grains are larger or less rounded than in an ordinary sandstone.

Hang-fire. An explosive charge which does not explode for some time after the fuse or straw has been lighted is said to hang fire.

Hauling Engine. An engine employed to move tubs on an underground engine-plane.

Head-gear. The frame carrying the winding pulleys erected over a shaft.

Heading, or **Headways.** An excavation driven at right angles to the bords.

Heaving. The floor of the mine " heaves " or " lifts " when it rises up in the excavations, owing to the pressure of the overlying strata upon the pillars of coal.

Holers. The men employed in the operation of holing the coal.

Holing. When one working-place meets another the opening is termed the " holing ", and the one is said to have " holed " into the other.

Igneous Rocks. Those rocks which have been subject to intense subterranean heat, and have cooled down from a state of fusion.

Intake. The road in the mine along which fresh air passes from the downcast shaft to the workings.

Jenkin. A narrow excavation driven through a pillar of coal.

Jig. A self-acting incline.

Joints. The natural cracks which occur in stratified rocks at right angles to the line of bedding. They are a very great aid to quarrymen and miners in detaching masses of rock.

Leg. A prop supporting the end of a cross-piece or crown.

Lifts. The slices or portions of a pillar of coal which are taken off in the removing of a pillar are termed " lifts ".

Lithology. That branch of geology which deals with the mineral composition and texture of rocks.

Longwall. A method of working coal by which the coal is extracted in one working.

Lye. A siding.

Main Air-ways. The roads along which the principal ventilating currents pass.

Main Levels or **Roads.** The first and leading excavations in mines which are made for the purpose of opening out or "winning" the coal, and which are intended to be the principal roadways of the mine.

Main Rope. The rope by which the engine hauls out from the workings to the shaft the train of full tubs.

Man-hole. A refuge hole formed at the side of haulage roads.

Miss-fire. An explosive charge which fails to explode after the ignition of the fuse is called a miss-fire.

Naked Light. A candle or any form of open light that is not a safety-lamp.

Old Workings. Workings in coal made at some former time from the same or some other shaft.

Onsetter. The man in charge of the shaft signals at the bottom of the shaft, and who changes the tubs when the cages arrive at the bottom.

Outburst. A sudden liberation of large quantities of gas.

Outcrop, Basset, or **Basset Edge.** The place where rock beds or coal-seams appear at the surface.

Packing. The stone required for building packwalls.

Packwalls, or **Packs.** Stone pillars or walls built between roof and thill on each side of a gateway.

Palæontology. That branch of geology which treats of the fossils which rocks contain.

Panel. A district of working-places separated by barriers from other districts.

Parting. The line of separation between two beds. Sometimes applied to a thin band of stone in a seam of coal, and termed a "dirt parting".

Pillar of Coal. A block of coal left standing to support the roof.

Pitbank, or **Pitheap.** The elevated stage around the top of a winding shaft upon which the tubs are delivered from the cages.

Pitwork. The pumps, appliances, and fixtures of a lifting or forcing sett in a pumping shaft.

Plans. Maps or drawings representing on a smaller scale the workings of the mine in their exact position and proper proportion.

Plunger. The solid ram of a force pump.

Props. Round pieces of timber set upright to support the roof.

Pulley-frames, or **Head-gear.** The wood or iron uprights erected at the top of winding shafts which carry the sheaves or pulleys over which the winding ropes pass.

Pumping Engine. The engine employed in raising water.

Pumping Shaft. The shaft in which the appliances are fixed for raising water from the mine to the surface.

Punch Props. Short props placed between each circle of cribs in a shaft.

Putter, or **Trammer.** A man or boy engaged in conveying tubs between a siding and the face of the working-places. A hand-putter or trammer is one who pushes the tubs by hand; a pony-putter is one who has charge of a pony engaged in putting.

Return. A road in the mine along which the impure air is conducted from the workings to the upcast shaft.

Rider. A guide frame for guiding a sinking kettle.

Rise. This is the opposite of " dip ". Workings going " to the rise " are going up the inclination or slope of the seam; those going " to the dip " are going in the opposite direction.

Roof, or **Top.** The bed of stone resting upon a coal-seam becomes the roof of the excavations made in the coal.

Roof-framy. A roof which is tenacious, and when allowed to fall breaks down in large blocks or " frames " of stone.

Safety Hook. A detaching hook to prevent overwinding.

Sandstone. Rock consisting of grains of sand cemented together. Sometimes termed " post ", " whin ", and " hazel " by miners.

Seam. A bed of coal is usually termed a seam of coal.

Shaft, or **Pit.** The opening made from the surface to the mineral which it is proposed to work or mine. Shafts are invariably made perpendicular in coal mining, but in metal mining they are sometimes inclined from the perpendicular.

Shaft Pillar. A large pillar of coal left around the shaft.

Shaft Siding. The station or landing-place arranged for the full and empty tubs at the bottom of the winding shaft.

Shale. The term shale is applied to all consolidated beds containing mud or clay in large proportions. The terms used by miners for various varieties of shale are: blue-stone, blue-metal, grey-metal, and plate.

Shot. An explosive charge.

Sinking. The process of opening out or " putting down " a shaft; the men engaged in the work are termed " sinkers ".

Sinking Kibble. A large bucket for raising the stones, &c., from a shaft being sunk. Sometimes called " bowk ", and " hoppett ".

Sleepers. Pieces of timber upon which rails are laid and secured.

Stall. A wide working-place in coal. In some districts, a bord.

Staple. A perpendicular stone drift in a coal mine is called a staple when it does not communicate with the surface. They are frequently made in mines to communicate between seams.

Stonehead. The first bed of rock reached after passing through the soil, clay, or sand at the surface.

Strata. Two or more beds together are termed strata.

Stratified. Rocks resting one above another in more or less parallel layers are said to be stratified.

Stratum. When rocks occur spread out in layers, beds, bands, or seams, each one forms a stratum.

Straw. Straw containing a train of powder sometimes used in the firing of shots.

Strike. A line across a series of strata at right angles to the direction of the dip.

Stringing Deals. Deals nailed to the inside of cribs in a shaft to support the whole of the timbering.

Stuffing-box. A cylindrical box filled with teased hemp at the top of the working barrel of a plunger.

Suction. The vertical distance between the working barrel of a pump and the sump, up which the atmospheric pressure forces the water.

Sump. A lodgment for an accumulation of water in the mine from which it is pumped.

Sumping Shot. A shot placed to blow out a piece of stone so as to unkey the bottom of the shaft for the other shots. A shot placed to unkey the face of a drift or working-place in the mine is also termed a sumping shot.

Tail Rope. The rope by which the engine hauls " inbye ", or from the shaft to the workings, the train of empty tubs.

Thick Seam. A seam of coal about 6 feet in thickness and upwards is usually termed a " thick seam ".

Thill. The floor of the seam.

Thin Seam. A seam of coal about 3 feet in thickness and under is usually termed a " thin seam ".

Tramway. A road laid with short light rails upon which the putters run the tubs.

Trap-door. A door for the purpose of guiding the ventilating air-current.

Under-clay, Fire-clay, or **Seggar-clay.** The bed of clay upon which coal-seams usually rest.

Under-cutting, Holing, or **Kirving.** The cutting out at the bottom or some other part of a seam of coal a wedge-shaped piece in order to facilitate the getting of the coal.

Underhand Work. Hacking or cutting the floor of the mine.

Unstratified. Rocks occurring irregularly, having no parallelism, are said to be unstratified.

Upcast Shaft. The shaft up which the air returns to the surface after having ventilated the workings of the mine.

Upthrow. A fault where the displacement of the seam is upwards.

Ventilating Currents. The currents of air travelling in mines.

Vitiated Air. Air which has been rendered impure by the breath of men and horses, or by being mixed with the various gases given off in mines. It is frequently called " return air ".

Wagon-way, Rolley-way, or **Horse Road.** The roads along which tubs are conveyed by horses.

Wall. A communication or holing made at right angles between two bords.

Water-bearing Strata. Beds which yield a large quantity of water when bored or sunk through are said to be water-bearing.

Whole Coal. A tract of solid coal which has not been entered by any mine workings; sometimes termed " virgin coal ".

Whole Workings. Workings entering into whole coal are termed " whole workings ". The leading workings are termed " winnings ".

Winding, Drawing, or **Hoisting Shaft.** The shaft used principally for raising coal from the mine to the surface.

Winding Engine. The engine employed to raise coal up a shaft.

Windlass. A machine used for raising weights, usually worked by hand.

Working-places. Excavations in the mine which are being driven or extended in coal.

Yardage. The price paid per yard for driving a roadway.

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